COAL DIVISION

1940

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COAL DIVISION
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FOREWORD

The first volume of the Coal Division was published just ten years ago. This volume is the seventh one published by the Division, and consists of various papers presented at the annual winter meetings of the Division in New York in 1939 and 1940, and the fall meetings held at Chicago in 1938 and at Columbus in 1939.

Four years ago the Coal Division at its fall meeting in Pittsburgh participated in a joint meeting with the Fuels Division of the American Society of Mechanical Engineers. The object of that meeting was to bring the consumers and producers of coal together for a discussion of their common problems. The problems of the producers and consumers of coal are interrelated to a large degree, and such joint meetings have been very helpful in bringing about a better understanding between the two groups. The meetings have proved so valuable that they have been continued since the time of the Pittsburgh meeting and the next joint meeting of these two groups will be held at Birmingham, Alabama, in November 1940.

Each year the Coal Division is expanding its sphere of usefulness to the coal industry. Starting as an infant Division of the Institute in 1930, it has grown rapidly in membership and importance.

This volume contains a wide diversity of papers dealing with sampling, production, preparation and utilization of coal. Together with the preceding volumes of the Division, it makes available to the members of the Coal Division the latest scientific information pertaining to the coal industry and records the scientific progress made by the industry during the past decade.

CHARLES E. LAWALL,
Chairman, Coal Division.

MORGANTOWN, W. VA.
August 9, 1940

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A Decade of Sampling

By E. S. Grumell*

(New York Meeting, February 1939)

The correct sampling of coal and coke is becoming important to an ever increasing number of producers and consumers. This, therefore, may be an opportune moment to examine where we stand with regard to existing and proposed methods. In this paper an attempt will be made to review, as briefly as possible, some of the considerable amount of painstaking research which has been done in the last 10 years, and which has unquestionably led to a much better understanding of the subject. In so doing we shall try, on the one hand, to make clear where substantial progress has been made, and on the other hand, to draw attention to points that require further investigation.

The paper is divided into sections in order to emphasize the major divisions of the subject; further, in order to retain this emphasis, detailed consideration of some of the points has been relegated to appendixes. No attempt has been made to give a complete survey of the literature, as this has already been done by L. A. Bushell.[2] The following abbreviations will be used: A.E., average error; P.E., probable error; S.W.R., size-weight ratio; B.S.I., British Standards Institution.

Section 1.—Cost

It has often been stated that the cost of sampling should bear some relationship to the value of the product, and since coal is a relatively cheap raw material it would be a major error to specify, for ordinary commercial purposes, elaborate and costly methods of sampling which nobody would adopt. Specifications must be of such a nature that they will be used. *We believe that one of the achievements of the last* 10 *years has been to reduce the cost of sampling.*

There is, however, another point to be considered; viz., that the cost of sampling should bear some relationship to the value of the investigation. Correct sampling is a cog (possibly a small one) in the wheel

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* Imperial Chemical Industries, Limited, London, England.

[2] References are at the end of the paper.

of modern competitive industry, and must function smoothly, and against the argument that it is not worth while to spend time and money on research into coal sampling it must be remembered that all the experience in this problem is applicable to other problems of sampling. There is also a growing demand for a much higher standard of sampling technique in connection with the many researches which are now being conducted into the preparation and utilization of coal, which will have to be provided for. Lastly, owing to the existence of reliable methods of sampling, it has been possible in at least one colliery to modify the system of mining in such a way that it has reduced the cost of production. We are not at liberty to discuss the modification, we can only say that it depends on reliable sampling.

Therefore, while it is still true that the cost of sampling coal should generally be kept as low as possible, it is no longer entirely true to say that it must be reduced to a minimum because coal is a relatively cheap raw material.

Section 2.—Degree of Similitude or Accuracy

Ten years ago nobody had the slightest idea of the degree of accuracy of a sample. We were told to take 1000 or 1500 lb. and that this would be "representative." But Webster's dictionary tells us that "representative" means "exhibiting a similitude." It does not stipulate the degree of similitude. Today the position has been completely changed. We are now in a position to state—with a reasonable degree of certainty—that if specified methods of sampling are adopted the result will be "representative" of the original within certain clearly defined limits.

The first standard adopted by the B.S.I. was such that if 100 samples were taken simultaneously by the same method, from a consignment, 99 of them would, when analyzed, give an ash content within ± 1 unit of the true ash content; that is to say, if the true ash content of the consignment were 10 per cent, the ash content of 99 of the samples would lie between 9 and 11 per cent. Unfortunately, this standard was misconstrued by many people, who thought it meant that the proposed method of sampling would, in general, give results differing from the real value by $\pm$ one unit in the ash figure; that is to say, when the ash was really 10 per cent, most of the samples would give either 9 or 11 per cent, and they considered this to be much too great a tolerance—a misconstruction which is quite easy to understand. Actually the standard is a very high one, and means that in the majority of cases (82 out of 100) the sample will lie within 9.5 or 10.5 per cent ash, and in 50 out of 100 (even chances!) will lie within 9.74 or 10.26 per cent, which, for ordinary commercial purposes, is surely adequate.

This standard means that out of 100 samples, on the average:

1 will deviate from the true value by more than	± 1.0	± 0.5
7 will deviate from the true value by not more than	± 0.7	± 0.35
12 will deviate from the true value by not more than	± 0.6	± 0.3
18 will deviate from the true value by not more than	± 0.5	± 0.25
50 will deviate from the true value by not more than	± 0.26	± 0.13

If a higher degree of similitude or accuracy be required, this can be obtained (with certain qualifications) by altering the method of sampling; for instance, if it be desired that 99 out of the 100 samples should not deviate by more than ± 0.5 (i.e., should lie within 9.5 or 10.5 per cent), the probability of what will happen is given in the bracketed column above. In this case, 50 per cent of the samples will be within ± 0.13, which is very nearly within the analytical error.

The modern method of expressing the probability of any degree of accuracy is of the greatest importance and, as will be shown in later sections, has been demonstrated to be correct.

Another point, which follows from this and which was not known 10 years ago, is that, in order to increase the accuracy from the first standard of the B.S.I.—that 99 in 100 samples should lie within ± 1.0 unit—to a similar probability of ± 0.75 or ± 0.5, the gross sample should be nearly doubled in the first case and quadrupled in the second case. To give a concrete example, supposing a sample of 200 lb. were required to give an accuracy such that the chances were that if 100 samples were taken, 82 of them would give results within ± 0.5 of the true value; if it were desired that 82 of them should give a result within ± 0.25, the sample would have to be 800 lb. Moreover, this would involve the assumption that no greater errors would be incurred in reducing the 800 lb. to 2 grams required by the laboratory than in reducing the 200 lb., which is a very doubtful assumption.

It is important to emphasize that exaggerated degrees of accuracy are almost impossible of attainment, irrespective of cost.

Before leaving the subject we should like to say that we feel sure that the modern way of expressing results as a probability is correct. We have, however, been led into a certain amount of confusion caused by the gradual development of the adoption of this method of expressing results. There are at present in existence three standards of probability; viz., the $^{90}/_{100}$, the $^{95}/_{100}$ and the $^{99}/_{100}$. We suggest that an early opportunity should be taken of deciding which of these standards is to be preferred.

Section 3.—The Purpose of Sampling

The degree of accuracy required, and hence the method of sampling to be adopted and the cost thereof, will depend on the purpose of the investigation, as follows: (1) sampling a consignment for special purposes;

(2) sampling a consignment for ordinary commercial purposes; (3) sampling a consignment for ordinary routine purposes. Provision is being made in specifications for 1 and 2, but, thus far, very little interest has been shown in 3, which is at least as important as 1 and 2.

All power stations and factories wish to check their deliveries either for maintenance of quality or to obtain an average value for a period of a week or a month in order to calculate their thermal efficiency or fuel consumption. A description of how this might be done at minimum cost was described in an earlier paper.[7] If technical managements are using the methods designed for 1 and 2 for period ascertainments, they are incurring an unnecessary amount of trouble and expense.

The purposes of sampling must be further subdivided according to the locality of the sampling point, for instance: (1) loading into and unloading from ships' holds; (2) chutes, conveyors, bucket and other elevators, etc.; (3) stationary wagons; (4) stocks.

Section 4.—What Properties of the Coal Should the Sample Represent?

The object of the sample is to provide a means whereby the properties of the original can be ascertained, such properties being ash content, ash-melting point, calorific value, volatile matter, sulphur, etc., together with the size analysis.

Ascertainment of the Ash Content Correctly Represents other Properties, except Moisture.—The ash percentage is usually taken as the criterion of sampling accuracy for coal,* and that this is justified—or that a correct ascertainment of the ash content indicates a correct ascertainment of other properties—is proved by Morrow and Proctor,[15] who show that sulphur content and ash-melting point are satisfactorily represented by samples representative of ash content, subject to the qualification that no quality can be correctly or quantitatively ascertained unless the water content of the coal has been correctly estimated. For some time G. B. Gould, who has a real knowledge of coal values, has fully appreciated this point and discussed it in his paper,[5] but it is only recently that others have begun to appreciate that calorific values are as much depressed by 1 per cent of water as by 1 per cent of ash, and that, in other directions, such as the power required for pulverizing, water is an objectionable diluent.

Special Sample for Moisture.—In order to obtain a correct measure of the original water content of the coal it is necessary to take a subsample from the gross sample and to deal with it in a special way. Exactly how this should be done is still a subject for further investigation.

* For coke, moisture is taken as the variable and criterion.[5,15,16]

Section 5.—Taking the Gross Sample

Fallacy of Large Gross Sample

For many years there was held a vague idea that a gross sample must weigh anything from 500 to 2000 lb., and that the larger it was the more representative it would be. This idea has been exploded by Morrow and Proctor,[15] who showed that samples weighing 30 to 40,000 lb. gave the same results as samples weighing 600 to 700 lb. It is very likely that still smaller samples would have given the same results. It is true that the larger the sample the more representative it will be, although, as already pointed out, the proportionality is not direct, but the real reason why the large sample is not most satisfactory is because it cannot be reduced to the size required for the laboratory without incurring appreciable errors or excessive cost, or both.

Application of Theory of Errors

Industrial coal is composed of hundreds of thousands of pieces, which differ in size, ash content and distribution, and before anything can be said about the weight of the gross sample, some means must be found for ascertaining how much they differ and how they are distributed.

Such a means is provided by the Theory of Errors, or the Theory of Probability. This theory has been explained in many papers, so that here only the simplest example of its application will be given:

Supposing 100 cars of coal are each sampled individually in such a way that the result is closely representative of the real ash content of each car; or supposing that 100 increments are taken from one car, and each treated similarly: then, in both cases, we shall have a series of 100 ash tests, the average of which is the ash content of the whole consignment or of the individual car. Now if the deviation of each individual test from the average be calculated and added up, irrespective of sign, and divided by 100, we shall obtain what is known as the Average Error, or by multiplying by 0.85, what is known as the Probable Error.

This function is a definite measure of the way in which the pieces differ in size and ash content and in distribution, or, in other words, is a measure of the heterogeneity or nonuniformity of the material.

Once this measure has been obtained it is possible by a very simple mathematical formula to find out how many increments must be taken in order to obtain a gross sample that will be representative within a required degree of accuracy as defined in section 2.

Concrete Proof of Correctness of Theory

Many people regard the application of statistical or mathematical theories to practical problems with suspicion; in regard to more advanced

statistics, one might possibly sympathize with this point of view, but the methods used by research workers on coal sampling are relatively simple. However, for those who may still have some doubts, we can say that in all the long series of experiments carried out in America, South Africa and Great Britain, the actual distribution of errors has invariably been found to agree closely with that demanded by theory. This, we think, is a very important point. To whatever the Theory of Errors has been applied, whether to sampling by increments or cars, whether to analytical records or other investigations, *the found distribution always agrees with the theoretical.* On the other hand, a careful examination of records does seem to indicate that the exceptionally big deviation occurs a little more often than it should. Morrow and Proctor point out that their series are not absolutely Gaussian and have a "tail" in the higher regions for ash and sulphur which we have been able to confirm. The too frequent occurrence of the big deviation may be due to this; at the same time, statisticians assert that while they are prepared to guarantee the average distribution—which as stated above, is well justified—they are not prepared to guarantee the frequency of the maximum.

This does not in the least detract from the value of the theory: it merely raises the point as to whether it is necessary to guard against the occurrence of a big deviation in important cases by taking a duplicate sample. It also indicates that it is preferable to judge the application of theory to practice by comparison of A.E. rather than by the occurrence of maximum deviations. Largely owing to lack of data many of the conclusions in connection with the theory of S.W.R. have been based on the occurrence of a maximum deviation, often in a very limited series of observations, and this has sometimes led to apparently abnormal results. We think that it is definitely preferable, if possible, to make comparisons on a basis of A.E.

Increment Sampling

The idea of taking samples by increments—i.e., that samples should be collected by a certain number of increments of a certain size, evenly spread over the consignment—was the immediate outcome of the application of the Theory of Errors, and is now generally applied to all sampling problems.

Gross Sample Independent of Size of Consignment

Several authorities make this statement and no one, thus far, has contradicted it. It is certainly correct in theory, and in practice is limited only by the practicability of evenly distributing the increments. With very large consignments, especially when carried in ships, it may be desirable to take more than one sample.

Number of Increments and Weight of Gross Sample

The number of increments depends on the A.E. If all coals as mined or as prepared by cleaning and sizing had the same A.E. the position would be very simple, and only one number would be required, but coals differ considerably in A.E. and there are at least four things that influence it: (1) For the same type of coal, the amount of refuse or the total ash; (2) the type of coal, inherent ash; (3) the size; (4) the level of control.

1. *Relationship between Average Error and Total Ash Content.*—This has been investigated by Bushell,[2] Gould,[5] Morrow and Proctor[15] and ourselves, and the results are shown in Fig. 1. Generally speaking, for the same type of coal, an increase in total ash content—which implies an increase in the refuse ash—may be taken to indicate a greater degree of heterogeneity, and is accompanied by a higher A.E. or P.E. In Fig. 1 it is shown that the P.E. increases with the ash content and that the relationship is almost identical for South African, American and British coals. Arising out of this relationship is the decision that the number of increments and therefore the weight of the gross sample must be increased as the ash content increases, which is now provided for in specifications by *dividing coals into groups according to ash content.* It will be shown that the present divisions are neither perfect nor universally applicable and could be improved. The important point to recognize is the principle of dividing coals into groups for sampling purposes.

2. *Relationship between A.E. and Type of Coal* (*See Appendix* 1).—It is almost universal practice to take a specific gravity of 1.6 as the dividing line between coal and refuse. Therefore, in what follows, the material which floats at 1.6 is defined as "coal" and the material which sinks at 1.6 as refuse, which will be mostly free shale. Further, inherent ash is defined as the ash content of coal floating at 1.6 and defines what is meant in this section by "type of coal."

The following discussion arises from the fact that investigators in America and South Africa have found their Average Errors to be lower than those originally given by us in B.S.I. No. 403 for British coals.

Since A.E. depends on the variation of ash in the individual pieces, it is obvious that if all the pieces had, say, 20 per cent of ash, there would be no A.E. and a single piece would be "representative": equally, it is obvious that if the ash content of the pieces varies from nearly zero up to 70 per cent or more, the A.E. will be big. The former does not exist in nature but the latter applies to the majority of British coals, and all other coals must lie somewhere between these two extremes.

British coals contain, on the average, about 4.25 per cent of inherent ash distributed in particles ranging in specific gravity from 1.28 to 1.60,

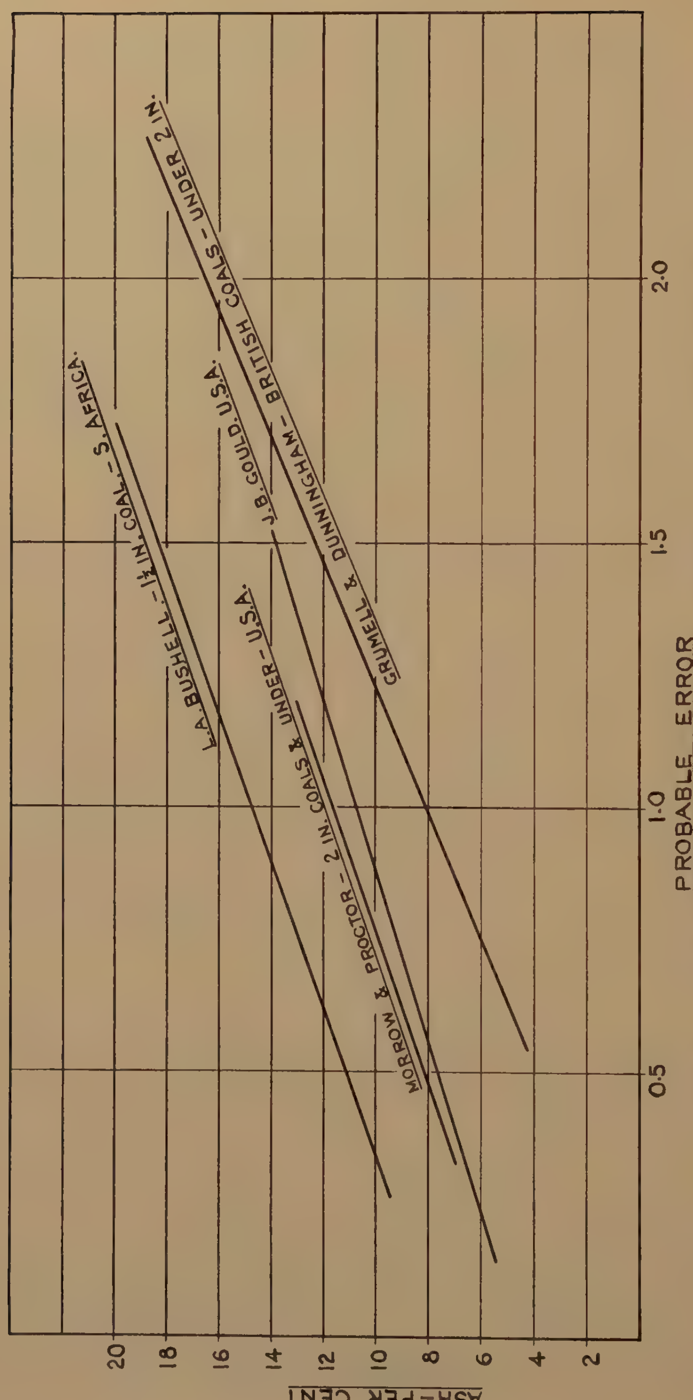


FIG. 1.—INCREASE OF PROBABLE ERROR WITH TOTAL ASH CONTENT FOR ANY GIVEN TYPE OF COAL.

and the refuse of these coals (in their raw state) contains 65 to 70 per cent of ash.

The South African coals investigated by Bushell contain, on the average, about 10 per cent of inherent ash, most of which is concentrated in the specific gravity limits of 1.35 to 1.60. The South African "coal" is therefore appreciably more "concentrated" than the British and will, therefore, be more uniform and have a lower A.E. Further, the refuse in South African coals has only about 50 per cent ash, so that the inclusion of refuse will have less effect with South African coals than with British coals.

It is, therefore, obvious that for the same percentage by weight of coal and refuse, the South African coal will be more uniform and will have a lower A.E. than British coals in spite of the fact that the *total ash* content of the South African coals is higher.

With regard to American coals, as far as we have been able to ascertain, there is a wide range of coals in the United States, some having inherent ash as low as British coals and some having inherent ash as high as South African coals. The *average* for American coals appears to be about 7.5 per cent. The ash in the refuse will vary and it is important to take this into consideration. It is also important to distinguish between cleaned coal and raw coal. The refuse retained in cleaned coals seldom contains more than 50 per cent of ash and is frequently as low as 35 per cent, whereas the refuse in raw coals may be as high as 70 per cent, so that 4 per cent by weight of refuse containing 35 per cent of ash from a cleaned coal will equal 2 per cent by weight of refuse containing 70 per cent ash in a raw coal.

The question arises: How can these coals be compared? Several formulas have been advanced by Bushell, Kassel and Guy, and others, which contain the fundamental principle that the A.E. is dependent on (1) the inherent ash and specific gravity concentration of the "coal" and on (2) the ash content of the refuse. They are, however, a little too complicated to be of universal application at present.

Failing a more correct formula, we have adopted a basis of "comparable inherent ash," which, while admittedly only an approximation, leads to the interesting result that American, British and South African coals, although differing in total ash content all have nearly identical A.E.'s.

This subject is so important that it is discussed in greater detail in an appendix, where a table is given showing the results referred to above.

3. *Relationship between A.E. and Size of Coal* (*See Appendix* 2, *p.* 29).—There are some indications that the A.E. increases with size; that is to say, that for the same ash content, a 2 to 0-in. slack or resultant will have a higher A.E. than ½ to 0-in. material. The evidence at present is incomplete.

4. *Relationship between A.E. and Level of Control* (*see Appendix* 3, *p.* 30).—Sometimes two coals of similar type and total ash content have different A.E.'s. This is quite probable and arises from different methods of preparation. With raw coals it may depend on the method of mining or the number of faces worked in the mine, or whether the coal is from one or more seams. With cleaned coal it may depend on whether it is from one or more seams, whether the preparation plant has a raw coal bunker or not, and (probably most important) on the technical control of the washing plant.

The idea of "level of control" is somewhat new, at least in its application to coal, and is more fully discussed in Appendix 3.

How to Find Average Error of Coal

The average error is usually ascertained by finding the ash content of individual carloads. The A.E. so found unavoidably includes sampling and analysis errors. It may also be found by taking a number of increments from a carload or from a stream, providing that the particular carload or stream chosen has an ash content equal to the average ash content of the consignment.

That both methods give similar results was indicated in our original publication (B.S.I. No. 403) and has since been confirmed by Morrow and Proctor in their investigation of coal B, when they obtain similar values by both methods.

Bushell points out that when duplicate determinations of the P.E. of the same variable have been made by Morrow and Proctor, the determinations show close agreement. In spite of this, we should like to revert to our original contention that in general a tolerance of ± 0.2 or ± 0.3 should be allowed in the determinations and that the A.E. should be regarded as an essential and most valuable guide in sampling problems, but that its limitations must be recognized.

Weight of Increment (*see Appendix* 4, P. 31)

The application of the theory of errors involves taking samples by increments, but the theory gives no guidance as to the weight of each increment. It seems reasonable to suppose that the increments should be approximately representative of their immediate surroundings both as to ash content and size distribution of particles.

It might be possible to calculate the required weight of increment mathematically, if all the necessary information were available. Also, the correct weight can be found empirically. This was done by Bushell for his particular coals, and his recommendations are in surprisingly close agreement with those contained in our original publication. In view, however, of the fact that the results were obtained with relatively

uniform South African coals, we are inclined to agree with the American view that the weight of individual increments should be slightly increased.

Practical Considerations

Even when the correct number and weight of increments have been determined, there remain many practical difficulties to be overcome, and to be explained in detail. Most of these details come under the heading that the point where the increment is taken must be free from errors caused by segregation or by bias.

The point is often raised as to whether the increment should be the complete cross section of a conveyor. This depends on how the conveyor is loaded. If the conveyor is evenly loaded, increments of the normal size can be taken at intervals across the conveyor from left to right, and vice versa. If the conveyor is unevenly loaded, a complete cross section must be taken, which will generally require a large increment, or a plate or other device must be inserted so as to give a uniform stream.

There are many other minor but important details to be considered, which would take too long to discuss in detail.

SECTION 6.—SUBDIVISION (SEE APPENDIX 5, P. 36)

It has taken a long time but at last it is being realized that the errors that may be incurred in subdividing a gross sample may be considerable, and may even exceed the errors involved in collecting the gross sample itself. Now that this has become recognized finality should soon be reached on this point, especially when the work reviewed in the appendix is considered.

It is my personal opinion that the procedure set out in the new tentative specification of the American Society for Testing Materials for subdivision of the gross sample may be an improvement on the latest B.S.I. specification. The following points might still receive consideration:

1. Whether it would not be desirable to specify two methods of subdivision, one for ordinary commercial purposes and another, more exact, for special purposes.

2. Whether—now that we no longer have to deal with gross samples of unmanageable weight—the gross sample could not be reduced in one operation to about 2 lb of B.S.I. 14 or Tyler 8 mesh. This might require a more efficient laboratory mill.

3. Whether, when a high degree of accuracy is required, it is desirable at some stage in the reduction to collect more than one subdivision.

4. In order to check up on errors, a little more information is required about small S.W.R. such as those used in the new A.S.T.M. specification. Unfortunately, much of the work done in the last two or three years has related to S.W.R. not usually obtained in practice.

The application of the Theory of Size Weight Ratio to subdivision is discussed in detail in Appendix 5, so no further reference will be made to it here. On the other hand, the following three points are of sufficient importance to be included in this section.

Coning and Quartering versus Riffling

We think that the old method of reducing samples by coning and quartering should be abolished or at least discouraged. C. W. H. Holmes[10] has shown that density segregation occurs, and some recent work in England, not yet published, has shown that variations in size analysis are greater with coning and quartering than with riffling.

Crawford's work[3] on the use of a riffle is of outstanding importance. In reducing 640 lb. of ¼-in. coal to sixty-four 10-lb. lots by riffling, he made a size analysis of a large number of subdivisions and found the deviations to be remarkably small. In fact, so good is the riffle that it can be used as a mixer. Crawford passed a synthetic mixture of stone and coal through a riffle three times and obtained the following ash percentages:

Determined	Calculated
23.9	23.7
31.6	31.3
46.6	46.1
52.3	53.3
67.7	67.6

Morrow and Proctor have also found the riffle to be very satisfactory. All the evidence is in favor of the riffle, and we think that coning and quartering might be abolished.

Sample-spout Crushers

Some crushers automatically divide the sample during the process of crushing. They are very useful but, unfortunately, unreliable. This statement has already appeared in print, and our own experience has been conveyed to the manufacturers. Some form of definite bias—probably formed by density segregation—occurs in these machines. Our own experience has been confirmed by a number of other users.

Errors Caused by Oversize

In crushing coarse coal or in grinding small coal to sizes required by the laboratory, any oversize particles usually consist either of the harder portions of coal or are unground particles of refuse. This has been stated by various authorities, and we think may now be accepted as an accurate statement. There is reason to suppose that many abnormal results either of analysis or sampling may be traced to this cause.

The effect of this grinding segregation has, so far as we know, not previously been taken into consideration, nor is it quite clear how one

should guard against it, but that it exists is clearly demonstrated by Crawford (p. 48, B.S.I. No. 763), who so arranged a milling operation that 6.9 per cent of oversize was obtained which contained 27 per cent of ash compared with 12.3 per cent for the remainder of the material. He repeated this experiment and obtained 9.2 per cent oversize with an ash content of 18.2 per cent compared with 8.7 per cent for the rest of the material.

We submit that this is a very important point, which has previously not been considered but which requires to be taken into consideration in future. It may explain many apparent anomalies.

Section 7.—Analysis (see Appendix 6, p. 45)

So long as the standard methods of determining the ash content are not beyond reproach, so long will all research work on subdivision of samples and collecting of gross samples be subject to appreciable analytical errors, and we are not satisfied that existing standards are beyond reproach.

They may be satisfactory for ordinary commercial purposes but even this is doubtful, and we submit evidence in Appendix 6 for a reconsideration of standard methods.

Section 8.—Checking

In the course of the last few years a considerable number of interesting and valuable researches on coal sampling have been presented, all of which are helpful in leading up to a standard specification for sampling. But the proof of the pudding is in the eating—that is to say that arising out of the research work certain definite proposals are put forward with regard to weight of sample and method of collecting and subdividing, etc. These proposals must then be submitted to confirmatory tests, or, in other words, having arrived at certain definite conclusions, they must be checked by taking from a consignment 10 or more samples according to the specification, and seeing whether the results conform to the specifications.

Conclusion

In this paper, especially in the appendixes, data published by other authors have been used, and we trust that they have been correctly interpreted. It is suggested that, in future, in investigations dealing with coal sampling, a short float-and-sink analysis should always be given.

In conclusion I should like to thank the Coal Division of the American Institute of Mining and Metallurgical Engineers for inviting me to give this paper, and to thank the Directors of Imperial Chemical Industries, Limited, for the permission to accept this valued invitation.

I should add that I am representing officially the British Standards Institution, which, in view of the fact that it now holds the Secretariat of the International Standards Association, is considering the revision of specifications for the sampling and analysis of coal and coke, including sampling for size analysis, and is attempting to draw up a specification which—at least in certain fundamentals—will try to meet with international approval. This opportunity of meeting our colleagues in America, where so much work of fundamental importance has been done, is therefore of the greatest value to us.

Appendix 1.—*Relationship between Absolute Value of A.E. and Total Ash Content, with Particular Reference to Type of Coal*

Originally it was assumed that the A.E. was in some way related to the total ash content, and that this assumption was correct is shown by the fact that *for a given type of coal* it increases with increasing total ash content. But this relationship does not hold for different types of coal.

It was T. W. Guy[9] who first pointed out that the absolute value of the A.E. was likely to be determined by the free ash content; i.e., the sinks at 1.6 sp. gr. This point has since been taken up by Bushell,[2] who has shown that it is partly proportional to the free ash and also to some extent on the distribution of the ash in the coal substance (i.e., the floats at 1.6).

That the A.E. should be primarily determined by the amount of refuse or adventitious matter (shale, etc.) mixed with the coal is fairly obvious on account of the high ash content of individual pieces, though this ash content may vary considerably; for instance, in South Africa the refuse in the raw coal seems to have about 50 per cent ash, whereas in Great Britain it is usually between 65 and 70 per cent. A distinction must also be drawn between the refuse from raw coal and the refuse retained in cleaned coal, the latter not infrequently having 20 per cent

Table 1.—*Ash Content of Various Specific Gravity Fractions*

Specific Gravity	British Coals, Per Cent	South African, Per Cent	Morrow and Proctor, U. S. A. Coal	
			Per Cent	Sp. Gr.
Floats at 1.28	1.7		2.9	
Floats between:				
1.28 and 1.30	2.8	2.9	4.9	1.28–1.31
1.30 and 1.35	5.4	5.5	8.3	1.31–1.34
1.35 and 1.40	9.9	8.8	11.8	1.34–1.37
			16.4	1.37–1.40
1.40 and 1.45	14.3	13.1		
1.45 and 1.50	19.3	17.7	23.0	1.40–1.60
1.50 and 1.60	25.3	22.5		
Total floats at 1.35 (weighted average)	2.8	4.8		
Total middlings 1.35 to 1.60	14.6	11.0		

less ash; for instance, 45 per cent would be a reasonable figure to take for refuse contained in cleaned coal and 65 per cent for refuse in the raw coal.

That the coal itself should also influence the A.E. will be evident from what follows.

For the same limits of specific gravity there is generally no great difference in ash content of British and South African coals, as is shown in Table 1, *but* there is an appreciable difference in distribution by weight, a much higher proportion of particles in South African coal being concentrated in the narrow limits of 1.35 to 1.6 specific gravity.

The great difference between two types of coal may be shown as in Table 2, using Bushell's sample A (p. 27, ref. 2*a*) as an illustration and

TABLE 2.—*Difference between Coals*[a]

	South African	British
Floats at 1.35	54.6 %[b] by weight, 5.6 % ash = 305	86 % by weight, 2.8 % ash = 240.8
Middlings between 1.35 and 1.60	45.4 % by weight, 15.0 ash = 681	14 % by weight, 14.6 % ash = 204.4
Total floats at 1.60	100 % 9.86	100 % 4.45

[a] Based on Bushell's sample A.

[b] 54.6 per cent by weight in sample A is rather high. The average given by Bushell in his Table 12 is only 33 per cent, which would give a still more concentrated coal.

comparison with British coal. These figures show that the inherent ash of South African coals is about 10 per cent compared with about 4.4 per cent for British coals (actual range 3 to 6), and that the South African coal consists of particles whose ash content does not vary as much as that of British coals, and is therefore likely to give a lower A.E. on the coal substance itself.

When it comes to including refuse, a similar but accentuated picture is shown. For instance, two coals of approximately 13 per cent total ash content would have the composition given in Table 3. The South

TABLE 3.—*Composition of Two Coals Having Approximately 13 Per Cent Ash*

	South Africa	British
Floats at 1.35	50 % by weight, 5.6 % ash = 280	75.6 % by weight, 2.8 % ash = 21
Middlings	41.4 % by weight, 15 % ash = 621	11.4 % by weight, 14.6 % ash = 166
Sinks at 1.60 (refuse)	8.6 % by weight, 50 % ash = 430	13.0 % by weight, 70 % ash = 910
Total	100 % 13.2	100 % 12.9

African coal is not only more uniform, but for the same total ash content contains a smaller percentage of refuse. For these reasons, for the *same*

total ash content, Bushell obtains lower A.E.'s than those given for British coals.

There is reason to suppose that the coals of the United States, *on the average,* lie about midway between those of Great Britain and those of South Africa (see following statement) with at one extreme, coals similar to the British, and at the other extreme, coals similar to the South African.

On the average, the inherent ash of coals is as follows: South Africa, 10 per cent; United States, 7.5; Great Britain, 4.3.

How is one to bring these to a common basis? Bushell,[2] Guy,[9] Dawe and Potter[4] have suggested formulas in their own investigations. I regret that I have not had time to investigate whether these can be applied universally.

We propose to adopt a method which we think gives a fairly close approximation. On the assumption that the A.E. is primarily dependent on the free ash content (which has been proved for S.W.R.), we propose to make a comparison based on the inherent ash.

In Table 4, the inherent ash of South African coal is 10 per cent and the inherent ash of the British coals quoted averages 3.3 per cent, a difference of 6.7 per cent. Therefore a South African coal of, say, 14 per cent total ash is equivalent to a British coal of 7.3 per cent total ash. Similarly, the inherent ash of Morrow and Proctor's coal A is 7 per cent and that of the British coals quoted 3.3 per cent, a difference of 3.7 per cent. Therefore coal A, with a total ash of 7.6 per cent is equivalent to a British coal of 3.9 per cent total ash. This is an approximation, and some allowance should be made for difference in size and for possible differences in coal concentration. When we compare coals on the above basis, the really striking fact emerges that investigators in South Africa, the United States and Great Britain find very similar values for the Average Error.

The A.E. of British coals in Table 4 was obtained from the commercial sampling of individual carloads over a period, in most cases, of 5 or 6 years and includes, in most cases, 500 to 600 carloads.

When it is considered the A.E. was originally intended as a guide, and is still, in our opinion, open to a tolerance of ± 0.3, the agreement shown in Table 4 is remarkable. The A.E. is unquestionably some function of the inherent ash and specific gravity concentration of the coal and of the amount of ash in the refuse—possibly also of size.

British coal No. 11 illustrates this point rather well. With 11.35 per cent total ash, it appears to be out of place in the table, but reference to the column "floats at 1.6" shows that it has an inherent ash of 5.8 per cent compared with the average of 3.3 per cent for the rest of the coals. On a "comparable inherent ash" basis, the "equivalent total ash" would be 8.8 per cent, which would place it in order. In this connection it is

TABLE 4.—*Average Errors of Coals on Comparable Inherent Ash Basis*

	British Coal						Morrow and Proctor, U.S.A. Coal, 2 In. and Under						Bushell, South African Coal,[b] 1½ In.				
	Coal	Size, In.	Floats at 1.6[a]	Sinks at 1.6[a]	Total Ash	A.E.	Coal	Floats at 1.6	Sinks at 1.6	Total Ash	British Equiv. Ash	A.E.	Floats at 1.6	Sinks at 1.6	Total Ash	British Equiv. Ash	A.E.
1	Washed Beans	⅜–⅛	96 at 2.1	4 at 49	4.0	0.43	A	98 at 7	2 at 39	7.6	3.9	0.46					
2	Washed Nuts..	1¾–1	99.5 at 2.7	0.5 at 39	2.85	0.44											
3	Washed Nuts..	1⅜–⅞	96.5 at 2.85	3.5 at 43	4.28	0.53											
4	Washed Pearls	9/16–¼	98 at 4.4	2 at 53	5.4	0.62	B	97 at 6	3 at 58	7.5	4.8	0.60	95 at 10	5 at 50	12.0	5.3	0.72
5	Washed Pearls	9/16–¼	96 at 3.8	4 at 41	5.3	0.69											
6	Washed Pearls	½–¼	96 at 2.5	4 at 54	4.6	0.70											
7	Washed Beans	⅞–⅜	96 at 4.3	4 at 47	6.0	0.77											
8	Dry Slack.....	⅞–0	92 at 3.7	8 at 65	8.6	1.08	1–6	89½ at 6.2	10½ at 45	10.3	7.4	0.94	90 at 10	10 at 50	14.0	7.3	1.04
9	Washed Slack..	¾–⅛	90½ at 3.9	9½ at 51	8.4	1.31											
10	Dry Slack.....	1½–0	92 at 2.85	8 at 66	7.86	1.33									16.0	9.3	1.40
11	Dry Slack.....	1¾–0	91 at 5.8	9 at 71	11.35	1.36											
12	Washed Slack..	⅝–0	90 at 5.2	10 at 45	9.1	1.38											
13	Dry Slack.....	1½–0	90 at 3.47	10 at 46	7.8	1.46											
14	Dry Slack.....	1½–0	91 at 2.1	9 at 56	6.9	1.48											
15	Dry Slack.....	1–0	84 at 3.8	16 at 57	12.3	1.64							80 at 10	20 at 50	18.0	11.3	1.74
16	Dry Slack.....	⅛–0	84 at 3.3	16 at 59	12.2	1.60											
	Average.....		3.3														

[a] 96 at 2.1 means 96 per cent by weight containing 2.1 per cent ash; 4 at 4.9 means 4 per cent by weight containing 4.9 per cent ash.

[b] Taken from Bushell, Fig. 3, p. 26.

also interesting to note that in Morrow and Proctor's Tables 6 and 7, two coals having A.E.'s of the order of 0.7 contain up to 6 per cent refuse, whereas two coals with A.E.'s of the order of 0.4 contain only up to 2.5 per cent of refuse.

TABLE 5.—*Washing Tests on American Coals*[a]

Coal from	Average Ash in Washed Coal, Per Cent	Number of Tests	Range of Coals Tested	Washing Tests
Illinois	8.07	11	5.77 to 9.37	A. In 1905
Indiana	8.2	7	6.15 to 9.83	
Ohio	7.2	8	6.03 to 8.57	
Pennsylvania	7.6	4	4.57 to 10.08	
West Virginia	5.5	5	3.47 to 7.76	
Arkansas	9.5	4	7.19 to 14.30	B. In 1906 and 1907
Illinois	8.5	18	6.50 to 10.26	
Indiana	7.5	2	7.09 to 7.85	
Missouri	9.9	3	9.45 to 11.05	
New Mexico	11.3	4	9.41 to 12.43	
Ohio	6.2	1		
Pennsylvania	6.6	4	5.30 to 8.02	
Tennessee	9.7	7	5.33 to 13.75	
West Virginia	5.3	2	4.87 to 5.76	

[a] Abstracted from U. S. Geol. Survey *Bull.* 336 (1907).

The figures in Table 5 show that there was close agreement between the results of washing tests in 1906 and 1907 and those of 1905 for the same states in the United States. It is therefore probable that the figures in Table 6 are representative of the ash content of coals washed in special tests.

TABLE 6.—*Special Washing Tests*

Coal from	Average Ash in Washed Coal, Per Cent	Number of Tests	Range of Coals Tested
Illinois	8.3	29	5.77 to 10.26
Indiana	7.9	9	6.15 to 9.83
Ohio	7.2	9	6.03 to 8.57
Pennsylvania	7.1	8	4.57 to 10.08
West Virginia	5.4	7	3.47 to 7.76
Arkansas	9.5	4	7.19 to 14.30
Missouri	9.9	3	9.45 to 11.05
New Mexico	11.3	4	9.41 to 12.43
Tennessee	9.7	7	5.33 to 13.75
Average	8.5	Total...80	3.47 to 13.75

In general, it may be assumed that in washing experiments of this nature not more than 2 per cent of refuse (sinks at 1.6) containing say 50 per cent ash would be left in the washed coal. This 2 per cent of refuse would account for approximately 1 per cent of ash. We should not be far wrong if we assumed that a laboratory float-and-sink test would indicate about 1 per cent less ash in the material floating at 1.6 than is shown in Table 6. Therefore we may assume that the inherent ash content of American coals floated at 1.6 sp. gr. is on the average 7.5 per cent, ranging from 4.4 per cent for West Virginia coals to 10.3 per cent for New Mexico.

The coals of the chief coal-producing areas (West Virginia, Pennsylvania, Illinois and Kentucky) contain on the average 4.4 per cent, 6.1 per cent, 7.3 per cent inherent ash (no value available for Kentucky). The inherent ash of British coals ranges for the various districts from 2.8 to 6 per cent and averages approximately 4.3 per cent, though approximately 50 per cent of the output is probably as low as 3.4 per cent. On the average, the inherent ash of American coals appears to be 3 per cent higher than that of British coals.

We believe, however, that there are a number of coals in America, and this is indicated in the preceding tables, with inherent ash as high as that of South Africa; for instance, the Roslyn bed, eastern Washington, has an inherent ash of 10.1 per cent (U.S. Bur. Mines *R.I.*3371) and another eastern Washington coal has 11.5 per cent inherent ash (*R.I.*3372). These coals, however, are not exactly like the South African, being distinguished as follows:

Specific Gravity	South Africa	R.I. 3371	R.I. 3372
Floats at 1.4 (on refuse-free basis)	53 % at 7 % ash	92 % at 10 % ash	93 % at 9 % ash

The coals investigated by Morrow and Proctor seem to be more like British coals, if we have interpreted their data correctly.

Appendix 2.—*Relationship between Absolute Average Error and Size*

On this point we have as yet very little positive information. Morrow and Proctor investigated coals 0–1⅛ in., 0–2 in., ⅜–1⅛ in., but do not appear to have obtained any significant difference. They also investigated 2–4-in. coal, and here it would appear as though the P.E. were higher. We certainly have indications that small fuels (½ in. down) have lower errors than, say, rough slacks 2 in. down (for same ash content), but this evidence is, as yet, incomplete.

APPENDIX 3.—*Effect of Control on Absolute Value of A.E.*

We have very definite evidence that the level of control influences materially the value of the A.E.

Taking, first of all, prepared coal—i.e., coal that has been passed through a cleaning, and possibly sizing, plant, for the same ash content and for the same type of coal, it is possible to have differing A.E.'s according to the level of control. (Not in one wagon so much as in the day's output.) Supposing the morning shift turned out coal with 8 per cent of ash, and the afternoon shift coal with 4 per cent ash, the average for the day would be 6 per cent, and the average error would be appreciable. Supposing both shifts turned out coal with 6 per cent of ash, the average for the day would still be 6 per cent and the average error practically nil (hypothetically).

A variation of this nature may occur not necessarily through inadequate technical control, but caused by variation in supply; for instance, when coal from more than one seam goes to the washery.

Table 7 gives a fair idea of variation over a period of years. Each of the fuels listed was tested by sampling about 100 individual carloads each year. Normal sampling procedure was adopted to comply with B.S.I. standard of ± unit in $^{99}\!/_{100}$. In each case the average ash content for any year was sensibly constant, not deviating by more than 0.6 from the average given.

TABLE 7.—*Average Error of Commercial Products*[a]
TESTED COMMERCIALLY BY CONSUMER ON SINGLE CARLOADS

Year	Coal No.									
	7	5	6	1	9	4	10	14	8	15
1930......			0.62					1.52	1.33	
1931......			0.51					1.54	1.27	
1932......		0.70	0.70	0.53		0.68	1.13	1.50	0.83	2.02
1933......	0.70	0.53	0.56	0.48		0.75	1.48	1.34	0.80	1.33
1934......	0.76	0.62	0.81	0.47	1.10	0.60	1.45	1.26	1.00	1.74
1935......	0.78	0.59	0.64	0.31	1.46	0.45	1.34	1.18	0.96	1.82
1936......	0.70	0.83	0.84	0.44	1.38	0.98	1.30	1.82	1.28	1.77
1937......	0.91	0.86	0.95	0.36		1.11	1.28	1.65	1.18	1.36
Average Ash.....	6.0	5.3	4.6	4.0	8.4	5.4	7.9	6.9	8.6	12 2

[a] The first six fuels are washed products of small size. The last four are dry, untreated slacks.

For the first five fuels, although there is some variation in A.E., it is sensibly constant. With the sixth fuel, something happened in the washery in 1936 and 1937 to upset the uniformity of the product.

The uniformity of a prepared coal depends on: (1) the technical control of the preparation plant; (2) whether the coal is from one or more seams; (3) whether there is a raw coal bunker before the plant.

With raw coal the uniformity depends firstly on whether the coal is from one or more seams, and further, on the method of mining. The influence of the former has been pointed out by Gould.

It is possibly due to variations caused by control that we originally stated—and still maintain—that a tolerance of ±0.3 should be allowed for the Average Error.

Addendum

Since writing the above remarks about control we have reread T. W. Guy's paper[9] and we are presenting one of our charts (Fig. 2), which differs from his because it shows a definite change in the level of control. One feels strongly tempted to write another appendix on the industrial value of these control charts, but that is not our subject. We must limit ourselves to the statement that the level of control influences the absolute value of the probable error of the coal. But apart from this, the paper by Guy contains information of the greatest interest. It confirms the general lot to lot variability of the coal, and I feel sure could provide much information about the actual average errors of various coals; it also supports the contention submitted in this paper for discussion that the errors of sampling, subdivision and analysis may outweigh the variability of the coal itself.

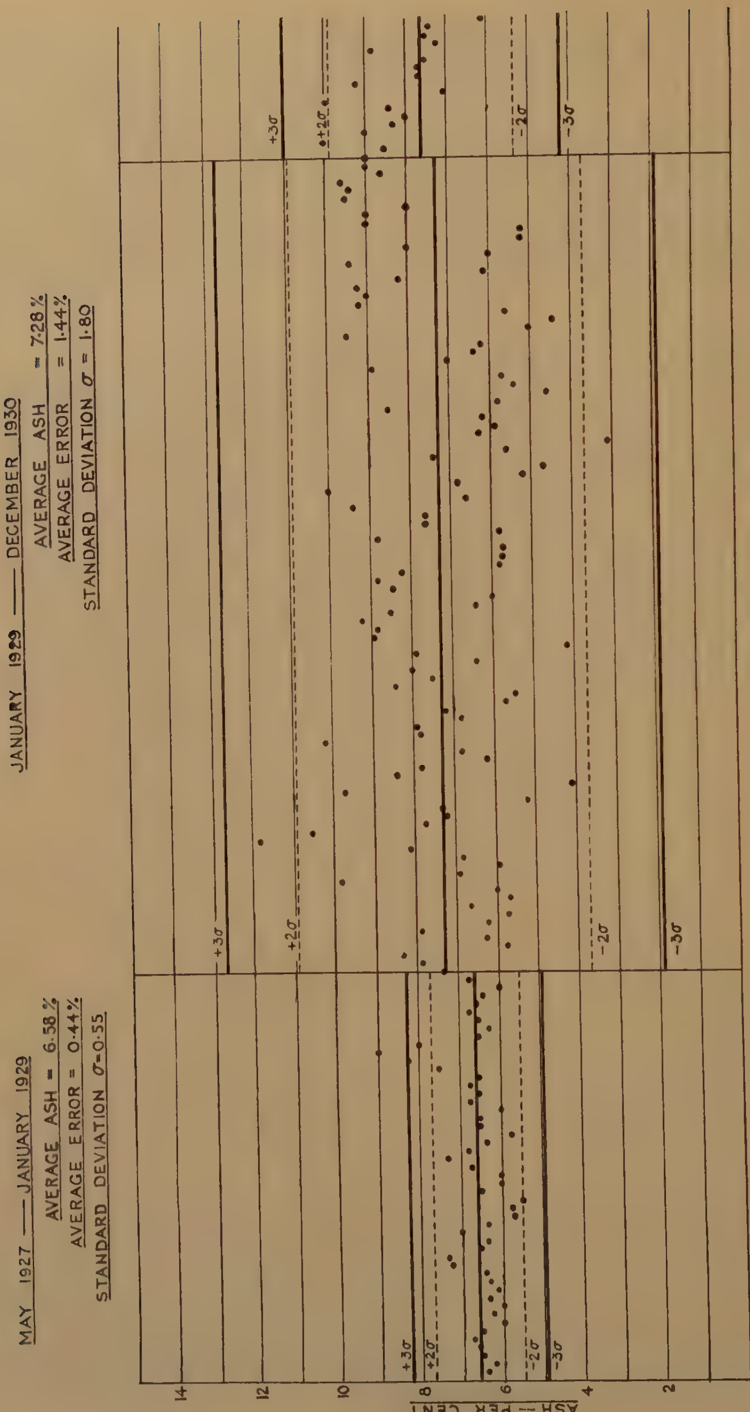
Appendix 4.—*Weight of the Increment*

The weights at present being recommended are given in Table 8.

Table 8.—*Weights Recommended for Increments*

British Coal	B.S.I. Specification No. 735, Lb.	B.S.I. Tentative Specification for Size Analysis (Unpublished), Lb.	A.S.T.M. Tentative Standard, Lb.	American Coal, In.
½-in	1	2	2	⅝
1-in	2	2		
1½-in	3	3	4	1¼
2-in	4	4	6	2
2½-in		6		
Large	10	10	10	2–6

The increment should be reasonably representative of its immediate surroundings both as to ash content and size distribution of particles. It is evident that if the increment is fairly small the chance inclusion or exclusion of one or more big pieces will alter its size analysis appreciably, and the chance inclusion or exclusion of one or more pieces of refuse will


MAY 1927 —— JANUARY 1929
AVERAGE ASH = 6·58%
AVERAGE ERROR = 0·44%
STANDARD DEVIATION σ = 0·55
JANUARY 1929 —— DECEMBER 1930
AVERAGE ASH = 7·28%
AVERAGE ERROR = 1·44%
STANDARD DEVIATION σ = 1·80
+3σ
+2σ
−2σ
−3σ
14
12
10
8
6
4
2
ASH — PER CENT

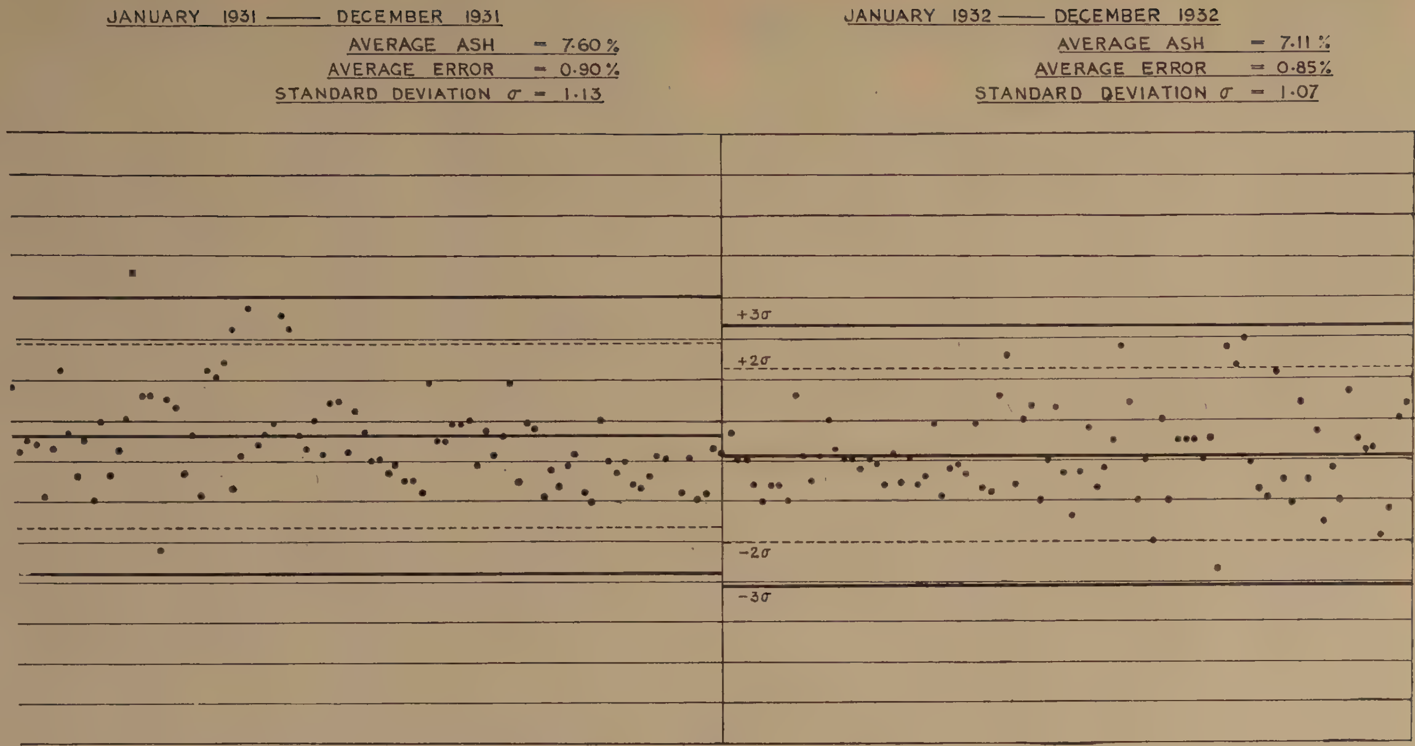


Fig 2.—Data on washed beans. Each dot represents the ash content of one carload. From Evaluation of Coal. Inst. Mech. Engrs. (1934). There is a slight overlap, left on the cuts for identification of position of spots.

alter its ash content. In both cases the increment fails to be reasonably representative and increments will differ greatly and lead to high and meaningless average errors. This is shown clearly in Morrow and Proctor's work where they went to the extreme of taking individual pieces of ½ to 1-in. coal and 1 to 2-in. coal, and obtained very large average errors.

In order that the increment may be reasonably representative, its weight should be varied according to the size of the coal, and, it now appears, according to the variability of the coal which, as stated elsewhere, is largely dependent on the percentage of refuse.

What the actual weight should be is still, for certain sizes, a little uncertain, but the work of Bushell in South Africa has indicated the method by which this can be settled, and for certain sizes has shown the weight to be taken. Before referring to this I should like briefly to refer to our original experiments (Tables 9 and 10).

Table 9.—*Experiments with Coal One Inch in Size*

Name of Coal	Oxcroft		Elsecar		Elsecar	
Number of increments	25	25	25	25	22	22
Weight of increment, lb	2	5	2	25	2	25
A.E. of increments	2.33	2.12	0.987	1.003	1.09	0.868
A.E. of corresponding cars	2.41		0.834			
Approximate ash test, per cent	16.5		8.0			

From experiments of this type we concluded that since increments of 2 lb., 5 lb. and 25 lb. gave similar average errors, which were in close agreement with the average errors obtained by sampling individual wagons, 2 lb. was a sufficient weight for coal 1 in. and less in size. This has subsequently been confirmed by Morrow and Proctor (ref. 15, Tables 24, 25, and 26), who obtain on this size identical A.E.'s with 2-lb. and 5-lb. increments.

Table 10.—*Experiments with Coal above One Inch in Size*

Coal	25 (2-lb.) Increments	25 (5-lb.) Increments
2 to 1-in. nuts, A.E.	1.80	1.22
2½ to 1¼-in. nuts, A.E.	1.76	1.25

Experiments with coal above 1 in. clearly indicated that for this size of coal a 2-lb. increment was not sufficiently representative (Table 10).

L. A. Bushell has made an exhaustive investigation into the relationship between A.E. and weight of increment, and has found, as we suspected, that it is represented by an asymtotic curve, which drops sharply

at the beginning and then flattens out. For 1½-in. coal this is shown in figures 1 and 2 (pages 22 and 24) of his paper.[2] Fig. 1 is reproduced here as Fig. 3. This being interpreted probably means that if the increment is a little too small the S.W.R. errors will be considerable, but once it reaches a certain weight—being a point on the flat part of the curve—any further increase in weight has very little effect on accuracy.

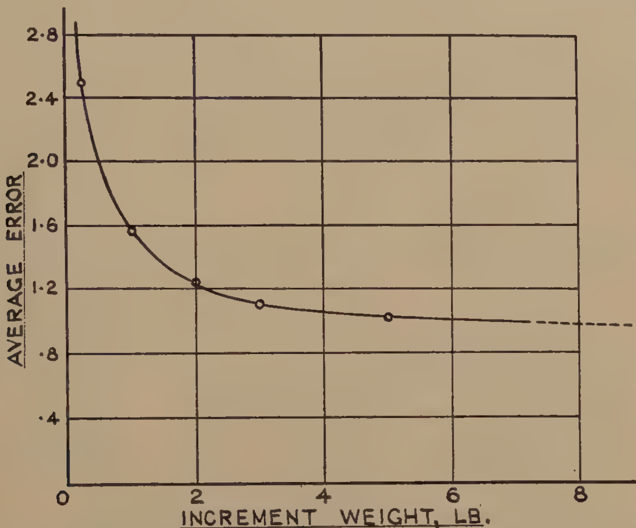


FIG. 3.—RELATIONSHIP BETWEEN A.E. AND SIZE OF INCREMENT.
From L. A. Bushell: The Sampling of Coal.[2]

It is difficult to overestimate the value of this work. From Bushell's Fig. 2—and still more if the graphs are drawn separately—it will be seen that the flat part of the curve starts a little later when the A.E. of the coal is high, indicating that a slightly greater weight of increment should be taken. His results for 1½-in. coal may be summarized as in Table 11. These results appear to be in close agreement with the recommendations

TABLE 11.—*Summary of Bushell's Data on 1½-inch Coal*

Coal	A.E.	Ash, Per Cent	Minimum Weight of Increment Sufficient Not to Alter A.E. Appreciably
C	0.52	12.3	2 lb.
J	0.54	10.5	3 lb.
B	0.68	11.5	2 lb.
E	1.02	12.2	
A	1.13	13.2	2 lb.
F	1.17	16.8	2 lb.
G	1.28	17.5	3 lb.
E	1.59	15.9	3 lb.
H	1.99	19.7	3 lb.

of the B.S.I., which stipulate a 3-lb. increment for $1\frac{1}{2}$-in. coal, and also warrant the previous assumption that 2 lb. is adequate for 1-in. coal. Recent British work on a specification for size analysis tends to confirm these conclusions.

On the other hand, it must be remembered that the South African coals are relatively homogeneous and further investigations may indicate the desirability of slightly increasing the minimum weights of increments for less uniform coals as has already been done by the A.S.T.M.

L. A. Bushell has extended his investigations to large coal and finds, like ourselves, that while there is some relationship between variability and ash content, it is not as precise as with smaller coal. His suggestions with regard to minimum weights of increments may be summed up as in Table 12.

TABLE 12.—*Bushell's Suggestions for Weight of Increment*

Coal	A.E.	Ash, Per Cent	Minimum Weight of Increment
J	1.06	9.91	12 or possibly 18 lb.
K	1.22	14.29	12 or possibly 18 lb.
L	1.57	12.44	12 or possibly 18 lb.
M	1.71	12.96	36 lb. Bushell's results on coal M are abnormal.

APPENDIX 5.—*Subdivision of Samples*

The importance of subdivision having at last been recognized, one may perhaps be forgiven for referring once again to Bailey's Theory of Size Weight Ratio, which, in our opinion, is still the best means of approaching the problem.

It is a little surprising that more than one author who has in recent years done work on the errors of subdivision has not converted his data into terms of S.W.R., and it leads one to suspect that either the necessary data are not readily available or that the application of the theory is not completely understood.

We therefore propose to give two examples of how it works, without explaining the theory, the figures being taken from B.S.I. No. 763, which contains the latest revision of the theory. The weights of particles to be used are listed in Table 13.

The latest A.S.T.M. specification recommends that the sample shall be crushed to pass a No. 4 sieve and shall then be divided down to 15 lb. = 15 × 453 grams. What errors are possible or probable? A No. 4 sieve has an opening of 0.185 in. and corresponds to the $\frac{3}{16}$-in. given in the table; the particle weight is, therefore, 0.074 and the S.W.R.

$$\frac{0.074 \times 100}{15 \times 453} = 0.00108$$

TABLE 13.—*Particles Used in Examples of Use of S.W.R.*

Square Mesh	Aperture of Major Mesh, In.	Average Weight of Particles, Grams
Inch:		
3–2	3	166
2–1½	2	68.6
1½–1	1.5	31.0
1–⅞	1	13.2
⅞–¾	0.875	9.0
¾–⅝	0.78	6.3
⅝–½	0.625	3.5
½–⅜	0.5	1.6
⅜–¼	0.375	0.5
¼–3/16	0.250	0.21
3/16–⅛	0.187	0.074
I.M.M.:		
⅛–5	0.128	0.032
5–8	0.100	0.014
8–10	0.063	0.004
10–20	0.05	0.0012
20–30	0.025	0.0001
30–40	0.0167	0.00006
40–60	0.0125	0.00002
60	0.0083	0.00001

The graph in Fig. 4 shows that if the coal contains 9 per cent of ash a subdivision of this size and weight will not deviate from the original in 90 times out of 100 by more than ± 0.125, or in 99 times out of 100 by more than ± 0.195.

The specification goes on to state that this 15 lb. shall be crushed to pass a No. 8 sieve and be reduced to not less than 1.75 lb. $= 1.75 \times 453$ grams. A No. 8 sieve has an opening of 0.095 in. and is, therefore, approximately equal to the 5 I.M.M. of the table, and the particle weight will be 0.014 and the S.W.R. $\dfrac{0.014 \times 100}{1.75 \times 453} = 0.00176$.

Fig. 4 shows that the subdivision will not deviate from the original in 90 times out of 100 by more than ± 0.155, or in 99 times out of 100 by more than ± 0.242.

The rest of this appendix reviews the work done by Bushell, by Dawe and Potter and by Crawford on subdivision, which, with the exception of some results obtained by Crawford, all tend to confirm the accuracy of the S.W.R. graph in Fig. 4. There are two points that require further consideration. It would seem as though the $^{90}/_{100}$ probability recommended in B.S.I. No. 763 should be increased to a $^{99}/_{100}$ probability, as the $^{90}/_{100}$ is frequently exceeded (possibly sometimes owing to analytical errors). The other point is that more information is required about the

errors attaching to small S.W.R.'s of the order referred to above, and we think that as a matter of practical policy this might receive immediate attention. We cannot progress any further with methods of obtaining gross samples of a high degree of accuracy until we have solved the problem of subdivision and of accurate analysis.

The graph in Fig. 4 is based on an inherent ash of 4 per cent, and was originally started from a basis of total ash content of 9 per cent. All other

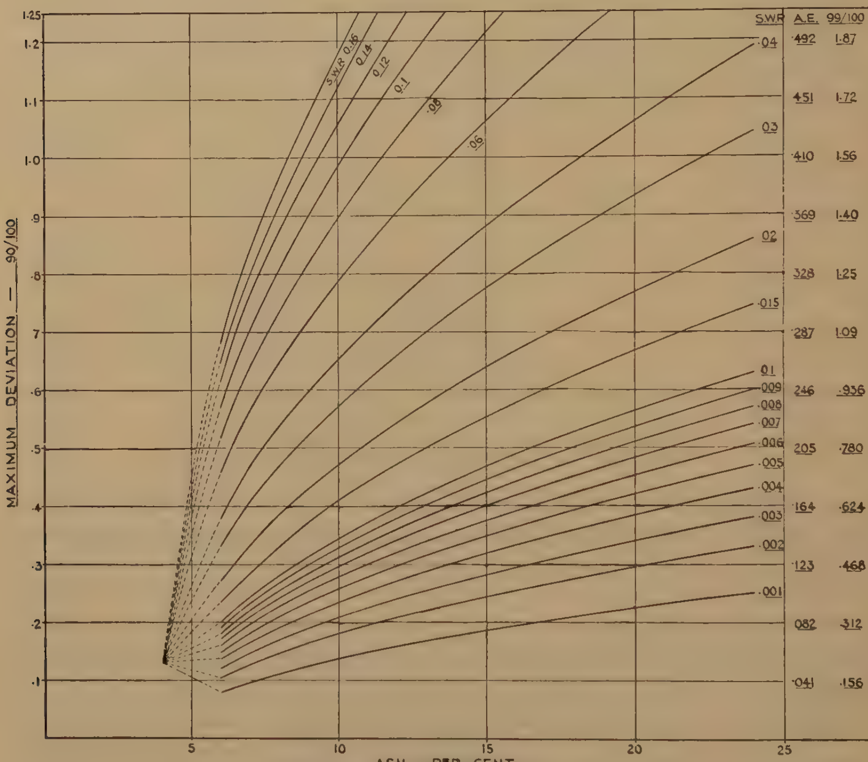


FIG. 4.—RELATIONSHIP BETWEEN ASH CONTENT, S.W.R. AND (1) AVERAGE ERROR OF SUBDIVISIONS, (2) MAXIMUM DEVIATION NOT EXCEEDED $^{90}/_{100}$, (3) MAXIMUM DEVIATION NOT EXCEEDED $^{99}/_{100}$.

From data in Table 5 of B.S.I. 763.

values have been calculated on the basis of the square root of the free ash content. Theoretically, therefore, the A.E. and deviation for a coal of total ash content of 4 per cent should be nil. This is obviously impossible because it neglects the variation of the ash in the coal itself, and the analytical error. Exactly how to deal with this is not clear, but we have adopted the ascertained fact that the maximum error of analysis for a refuse-free coal is not less than 0.13. All curves have, therefore, arbitrarily been drawn to this point.

Review of Parts IV and V of Bushell's Paper

Point 1.—The greater part of Bushell's work on this subject unfortunately deals with sample sizes which give S.W.R.'s outside the range usually employed in sampling and which, therefore, lose much of their value. In practice, Bushell's S.W.R.'s have to be avoided at all costs. However, his investigations yield results of great value.

Point 2. *Relationship between A.E. of Subdivision or Maximum Deviation and the S.W.R.*—It is a postulate of the Theory of Size Weight Ratio that the maximum deviation, or its equivalent the A.E., increases as the S.W.R. increases. Graphs showing this relationship have been presented before, the last being in B.S.I. No. 763. Bushell's data enable similar graphs to be drawn, and this has been done in Fig. 5, together with the corresponding graphs for the B.S.I. value. The relationship is sufficiently similar to indicate that this part of the theory of S.W.R. is reasonably correct and well established.

Point 3. *Relationship between S.W.R. and Ash Content.*—In Table 5 of B.S.I. 763, the maximum deviations for ash other than 9 per cent were calculated from the square roots of the free ash contents, at T. W. Guy's suggestion. This has been done in two instances where Bushell's data make it possible and assuming 10 per cent inherent ash in his coals (Table 14 herewith).

TABLE 14.—*Calculations from Bushell's Data*

	1. From Table 38, for ¼ to ⅛-in. Coal. S.W.R. 0.105			2. From Table 36 for ¼ to ⅛-in. Coal. S.W.R. 0.84. Taking Average of Q and T as Basis		
Colliery	P	R	O	Aver. of Q and T	R	O
Ash, total, per cent	12.26	18.05	24.0	12.0	17.5	24
Free ash, per cent	2.26	8.05	14.0	2.1	7.5	14
Maximum error: found	0.66	1.05	1.67	1.35	1.83	3.31
Calculated from square root of free ash		1.23	1.65		2.63	3.6
Average error: found	0.20	0.38	0.56	0.39	0.65	1.20
Calculated from square root of free ash		0.36	0.50		0.78	1.06

The agreement in the first case is perfect; in the second it is not quite so good for the maximum deviation of R but very good for the A.E., which is more important.

So this establishes the correctness of another assumption in connection with the Theory of S.W.R.

Bushell himself finds a direct relationship between the A.E. of subdivision and his factor F, which is a measure of ash content.

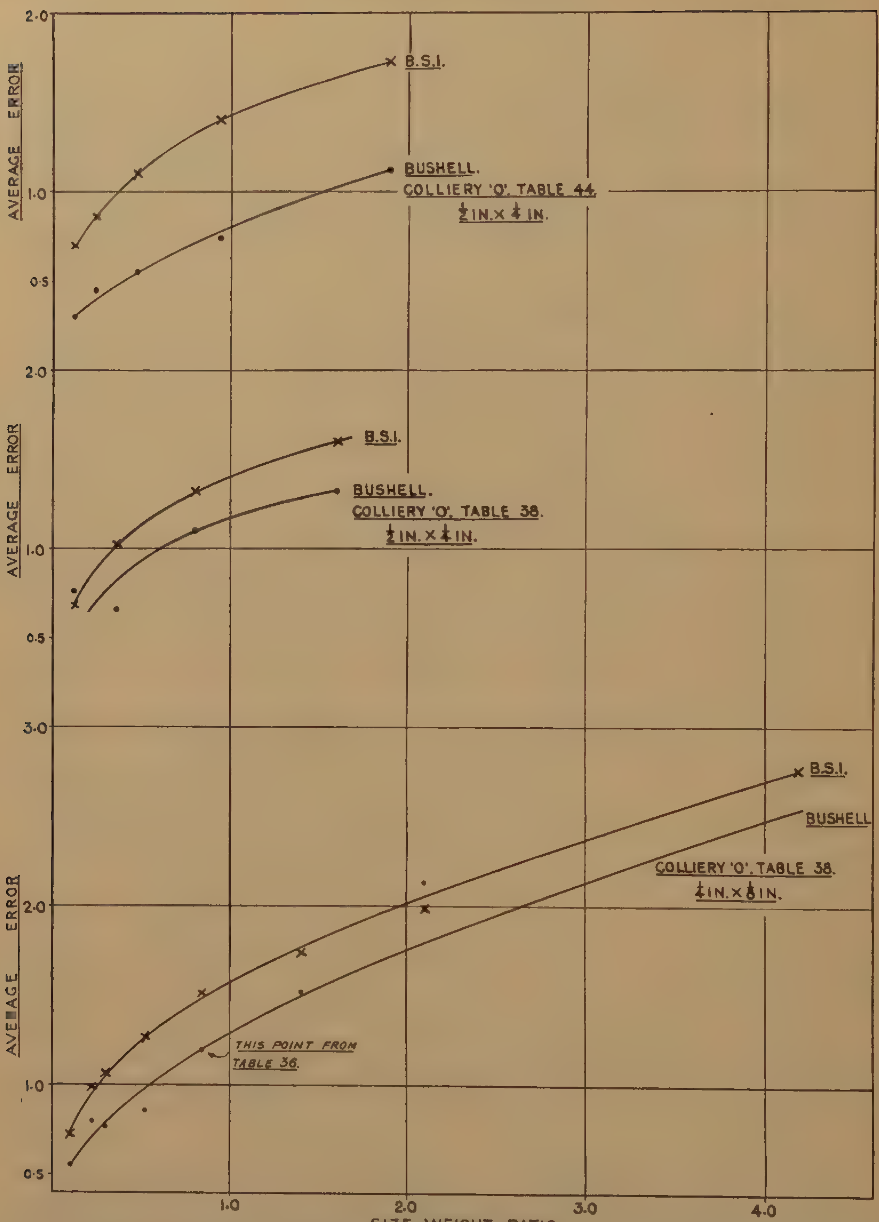


FIG. 5.—RELATIONSHIP BETWEEN AVERAGE ERROR, OR ITS EQUIVALENT MAXIMUM DEVIATION OF SUBDIVISIONS, AND S.W.R.

Point 4. *Does the Work of Bushell on Subdivision Confirm Magnitude of Errors Given in B.S.I.* 763?—The South African coals contain less sinks at 1.60 and higher inherent ash. We have, therefore, for comparison, equated them to British coals on a basis of inherent ash as explained in a previous section. Full details are given in Table 15.

The A.E.'s obtained by Bushell in his extensive investigations are in such close agreement with our development of Bailey's work that it is impossible to doubt the fundamental applicability of the Theory of S.W.R. Bushell's average errors are nearly always a little lower than ours, which, for the sake of safety, is satisfactory. That they are lower may possibly be explained by the greater homogeneity of the coal itself.

Point 5.—The maximum deviations found experimentally by Bushell exceed, in many cases, the values given in B.S.I. 763, which represent a $^{90}/_{100}$ chance, but they are nearly always within a $^{99}/_{100}$ chance (Table 15). In this respect Bushell's work confirms the findings of Crawford—and sometimes of others—in that the *maximum* does occur possibly more often than expected. It may be advisable to adopt, instead of Table 5 of B.S.I. 763, which has $^{90}/_{100}$, a table which has a $^{99}/_{100}$ chance. The graph in Fig. 4 gives both.

Point 6.—Calculated from Bushell's own probable errors, the number of times that the experimentally found maximum error in a series of 50 observations exceeds the calculated $^{99}/_{100}$ chance is a little disquieting, although several times the second value (Table 15) is appreciably lower and within the limit. Further, our examination of Bushell's data shows that the distribution of errors is statistically normal; i.e., what would be expected from the probable error.

Statisticians say that they are prepared to guarantee that *on the average* distribution should be accurately predictable, which has undoubtedly been proved, but that they are not prepared to guarantee when a maximum may occur. In the past, largely owing to lack of data, we have relied on the maximum deviation for our interpretation of S.W.R. theory rather than on the probable error. Now that the data are available, we should rely more on the probable error and less on the experimentally found maximum. At the same time, it seems necessary to guard against the occurrence of the maximum, which can only be done by arranging for the subdivision to be made in duplicate. This is definitely a point for serious consideration.

Point 7.—In Bushell's very few experiments with small S.W.R. of the order obtained in normal coal sampling, his results are definitely disquieting. (See first seven analyses of Table 15.) In these his A.E.'s tend to exceed those of the B.S.I., but not seriously. His found maximum deviations considerably exceed the standard B.S.I., even the $^{99}/_{100}$ chance, and either exceed or are very close to his own calculated $^{99}/_{100}$ chance.

TABLE 15.—*Summary of Bushell's Data*

Colliery	Total Ash, Per Cent	B.S.I. Equivalent	Weight of Sample, Grams	Size	S.W.R.	Found Maximum Error[a]	B.S.I. Maximum Error, 90/100	B.S.I. Maximum Error, 99/100	Bushell's Maximum Error, 99/100	Found A.E., Bushell	B.S.I. Average Error
					SUMMARY OF BUSHELL'S TABLE No. 38						
O	21.5	15.5	10	−20+30	0.0025	0.68 (0.59)	0.27	0.42	0.71	0.22	0.13
O	21.2	15.2	25	−20+30	0.0014	0.55 (0.54)	0.19	0.296	0.68	0.21	0.09
O	21.6	15.6	100	−20+30	0.00025	0.35 (0.27)				0.10	
O	21.9	15.9	100	−10+20	0.0055	0.72 (0.55)	0.38	0.59	0.68	0.21	0.18
O	21.8	15.8	200	−10+20	0.0027	0.51 (0.32)	0.28	0.44	0.49	0.15	0.135
O	22.5	16.5	100	−⅛−⅒	0.032	1.47 (1.36)	0.84	1.31	1.59	0.49	0.4
O	23.0	17.0	200	−⅛−⅒	0.016	1.28 (1.17)	0.62	0.96	1.24	0.38	0.3
O	23.7	17.7	5	−¼+⅛	4.2	7.48 (6.60)	5.7	8.85	7.40	2.28	2.73
O	23.4	17.4	10	−¼+⅛	2.1	5.97 (4.41)	4.2	6.55	6.95	2.14	2.0
O	24.6	18.6	15	−¼−⅛	1.4	7.19 (4.86)	3.6	5.62	4.92	1.52	1.73
O	23.8	17.8	40	¼−⅛	0.52	3.15 (2.58)	2.64	4.12	2.79	0.86	1.27
O	23.6	17.6	70	¼−⅛	0.30	2.98 (1.89)	2.21	3.44	2.50	0.77	1.06
O	23.3	17.3	100	¼−⅛	0.21	2.05 (1.91)	2.00	3.13	2.59	0.80	0.97
O	24.0	18.0	200	¼−⅛	0.105	1.67 (1.55)	1.51	2.36	1.81	0.56	0.73
O	26.7	20.7	25	½−¼	6.4	7.54 (6.83)			7.9	2.43	
O	26.4	20.4	100	½−¼	1.6	4.27 (3.71)	4.0	6.25	4.9	1.52	1.93
O	25.3	19.3	200	½−¼	0.8	3.42 (3.42)	3.19	4.98	4.1	1.26	1.54
O	21.3	15.3	100	½−¼	1.6	3.29 (3.05)	3.33	5.20	4.28	1.32	1.60
O	21.0	15.0	200	½−¼	0.8	3.32 (2.76)	2.74	4.27	3.56	1.10	1.32
O	21.6	15.6	450	½−¼	0.36	1.85 (1.55)	2.13	3.33	2.11	0.65	1.05
O	21.1	15.1	1360	½−¼	0.12	2.32 (1.67)	1.43	2.23	2.48	0.77	0.69
P	12.4	6.4	100	¼−⅛	0.21	0.88 (0.60)	0.75	1.17	0.81	0.25	0.36
P	12.3	6.3	200	¼−⅛	0.105	0.66 (0.56)	0.58	0.90	0.65	0.20	0.28
R	18.0	12.0	100	¼−⅛	0.21	1.47 (1.32)	1.50	2.34	1.52	0.47	0.72
R	18.0	12.0	200	¼−⅛	0.105	1.05 (1.02)	1.14	1.78	1.23	0.38	0.55
R	21.7	15.7	100	½−¼	1.6	3.91 (3.77)	3.32	5.20	4.15	1.28	1.60
R	20.7	14.7	200	½−¼	0.8	3.76 (2.78)	2.74	3.25	3.36	1.04	1.12
					SUMMARY OF BUSHELL'S TABLE No. 36						
O	24.0	18.0	25	¼−⅛	0.84	3.31 (3.18)	3.15	4.9	3.9	1.20	1.51
P	12.7	6.7	25	¼−⅛	0.84	1.87 (1.29)	1.38	2.15	1.43	0.44	0.57
Q	11.9	5.9	25	¼−⅛	0.84	1.58 (1.17)	1.18	1.84	1.46	0.45	0.57
R	17.5	11.5	25	¼−⅛	0.84	1.83 (1.49)	2.30	3.6	2.10	0.65	1.11
S	9.4	3.4	25	¼−⅛	0.84	1.28 (1.12)			1.11	0.34	
T	12.1	6.1	25	¼−⅛	0.84	1.12 (0.99)	1.18	1.84	1.07	0.33	0.57
					SUMMARY OF BUSHELL'S TABLE No. 44						
O	21.2	15.2	3	½−¼	0.118	1.16 (0.79)	1.43	2.23	0.97	0.30	0.7
O	21.2	15.2	1.5	½−¼	0.236	1.66 (1.29)	1.79	2.8	1.46	0.45	0.86
O	21.1	15.1	0.75	½−¼	0.472	1.67 (1.40)	2.30	3.59	1.78	0.55	1.1
O	21.2	15.2	0.38	½−¼	0.944	2.41 (2.16)	2.89	4.5	2.6	0.74	1.4
O	21.5	15.5	0.18	½−¼	0.888	3.27 (2.68)	3.58	5.6	3.64	1.12	1.73
					SUMMARY OF BUSHELL'S TABLE No. 46						
O	19.1	13.1	½	¼−⅛	0.092	0.69 (0.53)	1.15	1.8	0.84	0.26	0.56
O	18.7	12.7	½	⅛ to 10 mesh	0.014	0.49 (0.47)	0.5	0.78	0.55	0.17	0.24

[a] The figures in parentheses represent the value following the maximum.

The above must be qualified by the fact that Bushell describes the sieves used as being ordinary cement sieves. We may, therefore, have been wrong in assessing their apertures. It is most important that investigators should give accurate information about their sieves.

Review of Dawe and Potter's Paper[4]

Point 1.—This paper contains data of much value, especially as it includes some fairly low S.W.R.'s, though it may be pointed out that it is desirable in any future work to concentrate on still smaller values. Any subdivision which gives a S.W.R. of more than something of the order of 0.01 is for *practical* purposes not of great value.

Point 2.—The probable errors found by Dawe and Potter are in surprisingly close agreement with those of Table 5 of B.S.I. 763. With one exception they are nearly all slightly lower.

Point 3.—The maximum deviation found by Dawe and Potter exceed in five cases the values given in B.S.I. 763, Table 5, which represent a $^{90}/_{100}$ chance, but only once do they exceed the $^{99}/_{100}$. This tends to support the contention that a $^{99}/_{100}$ chance should be taken as standard in preference to $^{90}/_{100}$.

TABLE 16.—*Summary of Findings of Dawe and Potter*

Ash, Per Cent	Sample Weight, Grams	Size, In.	S.W.R	Maximum Deviation Found	B.S.I. $^{90}/_{100}$	B.S.I. $^{99}/_{100}$	Dawe $^{90}/_{100}$	Dawe $^{99}/_{100}$	B.S.I. P.E.	Dawe P.E.	Weight of Particle, Gram
12.95	55.6	1/4	0.38	2.03	1.81	2.84	1.78	2.82	0.74	0.735	0.21
12.45	42.4	1/8	0.075	1.21	0.98	1.53	0.81	1.27	0.40	0.334	0.032
12.53	29.0	1/2	0.052	0.54	0.84	1.31	0.41	0.64	0.34	0.167	0.015
12.35	20.0	1/20	0.006	0.28	0.32	0.50	0.22	0.35	0.13	0.092	0.0012
12.34	77.6	1/20	0.0015	0.12	0.18	0.28	0.14	0.21	0.074	0.056	0.0012
14.44	26.3	1/8	0.022	1.69	0.63	0.98	1.33	2.04	0.26	0.544	
14.00	61.0	1/8	0.052	0.76	0.93	1.45	0.66	1.04	0.38	0.273	
14.06	43.0	1/12	0.035	0.55	0.79	1.23	0.46	0.72	0.32	0.190	
13.84	31.0	1/20	0.004	0.42	0.30	0.47	0.28	0.43	0.12	0.114	
13.80	15.4	1/20	0.0078	0.38	0.40	0.62	0.36	0.56	0.16	0.147	
13.58	59.4	1/20	0.0022	0.16	0.23	0.36	0.18	0.28	0.09	0.074	
6.22	39.0	1/8	0.082	0.95	0.53	0.82	0.61	0.95	0.22	0.25	
				0.53							
6.13	29.0	1/12	0.052	0.34	0.41	0.64	0.26	0.41	0.17	0.107	
6.06	52.0	1/12	0.029	0.25	0.33	0.51	0.21	0.33	0.13	0.086	
9.40	15.4	1/8	0.21	0.72			0.54	0.84		0.22	
9.54	35.4	1/8	0.09	0.39			0.41	0.65		0.17	
9.45	24.0	1/12	0.063	0.15			0.15	0.24		0.064	
9.38	47.0	1/2	0.032	0.20			0.15	0.24		0.063	
3.88	59	1/4	0.356	0.95	0.88	1.37	0.74	1.15	0.36	0.303	
3.57	26	1/8	0.12	0.69	0.61	0.95	0.47	0.74	0.25	0.193	
3.63	19.2	1/12	0.078	0.25	0.51	0.79	0.24	0.37	0.21	0.098	
3.57	37.0	1/12	0.041	0.15	0.37	0.58	0.14	0.24	0.15	0.061	

Point 4.—In no case does the experimental maximum deviation exceed the $^{99}/_{100}$ chance calculated from Dawe and Potter's own probable error.

In general, their experiments are in close agreement with the B.S.I. standard (Table 16). In the text, the coal in the last test is said to have an inherent ash of 1.7 per cent, and would therefore be equivalent to a B.S.I. of 6 per cent ash in Table 5 of B.S.I. 763.

Work of Morrow and Proctor

Morrow and Proctor reported 256 subdivisions of a coal of ⅜-in. size containing 6.75 per cent ash, and a probable error of 0.094 obtained. Without knowing the actual size of the coal and the inherent ash, the B.S.I. equivalent cannot be calculated, but at a rough estimate, the probable error is of the order of 0.1, which is satisfactory agreement.

Review of Crawford's Paper

Crawford investigated the subdivision of a coal fairly high in ash (about 3 per cent inherent), containing about 2 per cent of sulphur and up to 4 per cent of carbonates, and obtained considerably greater deviations than the B.S.I. standard.

In reducing 640 lb. of ¼-in. coal to sixty-four 10-lb. lots, he found deviations exceeding the B.S.I. $^{9}/_{100}$ standard and also the $^{99}/_{100}$ standard, and the probable error calculated from his own data is quite appreciably higher than the standard, but his deviations are within the $^{99}/_{100}$ chance calculated from his own probable error, and his distribution of deviations is normal.

TABLE 17.—*Summary of Crawford's Data*

Ash, Per Cent	Sample Weight, Lb.	Size	S.W.R.	Found Maximum Deviation	B.S.I. $^{9}/_{100}$	B.S.I. $^{99}/_{100}$	Crawford		B.S.I. P.E.	Weight Taken for Analysis, Grams
							$^{99}/_{100}$	P.E.		
		Inch								
11.08	5	¼	0.0092	0.67	0.354	0.552	0.83	0.218	0.145	5
13.15	10	¼	0.0046	0.91 (0.66)	0.301	0.469	0.67	0.175	0.124	5
18.84	10	¼	0.0046	0.79	0.39	0.61	0.94	0.245	0.16	5
		BS								
11–15	⅝	14	0.0015	0.39	0.194[a]	0.302[a]	0.52	0.136	0.79[a]	5
19.1	⅝	14	0.0015	0.48	0.25	0.39	0.65	0.168	1.02	5
12.37	⅝	14	0.0015	0.43		approx 0.3		0.135		1, lid off
12.73	⅝	14	0.0015	0.40		0.3		0.146		1, lid on
12.77	⅝	14	0.0015	0.29		0.3		0.102		3, lid off
12.37	⅝	14	0.0015	0.28		0.3	0.46	0.122		6 ash tests

[a] For 13 per cent ash.

In reducing 10 lb. to ⅝-lb. lots, after grinding to pass B.S. 14 (aperture 0.047 approximately equivalent to Tyler 14) he obtained similar results.

This was confirmed in one instance by passing a 10-lb. lot to our laboratory, where similar results were obtained. It is difficult to explain this departure from the standard. The results are summarized in Table 17. In one respect the results are standard, in that for the first three coals the P.E. increases with the ash content approximately as the square root of the free ash:

Ash Content, Per Cent	Probable Error Found	Calculated from Square Roots
11	0.158	
13	0.175	0.175
19	0.245	0.22

Appendix 6.—*Determination of Ash Content*

The workers named in Table 18 have investigated the accuracy of the determination of ash content. For relatively high-ash coals of the order of 13 per cent, the probable error is of the order of 0.065. This implies that the analyses should show the following results: 68.6 per cent of the analyses should fall within ±0.097 of the mean value, 95.6 per cent of the analyses should fall within ±0.2 of the mean value, 99.3 per

Table 18.—*Determination of Ash Content*

Workers	Ash, Per Cent	P.E.	Reference
Davids and Fairchild	13.6	0.067	U. S. Bur. Mines *Tech. Paper* 171 (1918)
E. G. Bailey	11.5	0.07	*Jnl. Ind. and Eng. Chem.* (1909) **161**
L. A. Bushell	14.9	0.053	Fuel Research Inst., Pretoria, S. Africa. *Bull.* No. 8
L. A. Bushell	13.8	0.081	
Martin and Mandel	10 to 15	0.068	Amer. Soc. Mech. Engrs., Chicago, Feb. 1931
Crawford	13.1	0.06	British Standards Institution, No. 763, 1937
Crawford	14.4	0.042	
Imperial Chemical	14.5	0.076	Unpublished
Average		0.065	
Davids and Fairchild	8.9	0.043	
Morrow and Proctor	6.8	0.043	*Trans.* A.I.M.E. (1936) **119**, 227
Briscoe, Jones and Marson	3.7	0.035	Fuel Research Board, *Bull.* 29 (1933)
Imperial Chemical	1.4	0.025	

cent of the analyses should fall within ±0.26 of the mean value, the differences being correspondingly greater.

For lower ash coals with a probable error of 0.04, the results should be: 68.6 per cent of the analyses should fall within ±0.06 of the mean value, 95.6 per cent of the analyses should fall within ±0.12 of the mean value, 99.3 per cent of the analyses should fall within ±0.16 of the mean value.

These values seem to be reasonably satisfactory, but it must be noted that each was determined in one laboratory, probably with special care.

Morrow and Proctor state that Martin and Mandel found a maximum difference on the same sample of 2.15 per cent at different laboratories. Bushell states that he got a higher probable error of 0.081 (see above) with the 13.8 per cent ash coal because of spurting of coal from the ashing dishes during removal of volatile matter, in spite of the slow rise in temperature. Crawford (Table 4, p. 51, B.S.I. 763) obtains an ash content of 12.73 when the determination is done with the lid on, and 12.37 with the lid off, a difference of 0.36 in average ash content. Imperial Chemicals, in a similar investigation, found that with the lid on the ash content was 0.43 higher. Moreover, the probable error of the series with lid on was 0.076, but with the lid off it increased to 0.289. Crawford also shows that the weight taken influences the result, for instance:

	Gram	Ash, Per Cent	Gram	Ash, Per Cent	Difference
Table 4................	1	12.37	3	12.77	0.4
Table 8................	2	13.16	5	13.33	0.17

Morrow says that the probable error of an ash analysis can be materially reduced by burning 2 grams of pulp rather than 1 gram.

Konejung[13] passed the same sample to 16 laboratories and obtained a probable error of 0.33 which means that only 68.6 per cent were within ±0.5 of the average and 95.6 per cent within ±1.0, the difference probably being nearly double.

Dr. King[12] passed samples to 14 or 15 laboratories, where, in each case, determinations were made at least in duplicate, with the following results:

Average	Maximum	Minimum	Maximum Deviation	Maximum Difference
5.51	5.60	5.34	0.17	0.26
5.44	5.70	5.2	0.24	0.50
6.05	6.2	5.8	0.25	0.4
1.85	2.1	1.4	0.45	0.7

The specified methods for the determination of ash need reconsideration, in view of the appreciable differences which may occur between different laboratories. Even in the same laboratory the fact that differences up to 0.5 can occur indicates that the method is not sufficiently satisfactory or that reliance cannot be placed on a single determination. This may not be of great importance in ordinary commercial practice, but when the ash content is used in the application of the Theory of Errors or in the Theory of S.W.R., there must be some means of determining it accurately.

In the last British specification for the purpose of efficiency tests, weights of gross samples were specified for an accuracy of 0.25 to be obtained $^{99}/_{100}$ times. At present this is only just within the tolerance of the analysis!

That the analytical errors can have an important bearing on research is illustrated in Table 4 of Crawford's paper, from which the following figures relating to the variability of 16 subdivisions of a 10-lb. sample are taken:

Error	Based on a Single Ash Determination	Based on Six Ash Determinations
Maximum difference between subdivisions	0.89	0.48
Maximum deviation of any one subdivision from the mean	0.50	0.28
Probable error of the series	0.157	0.122

Nearly half of that maximum deviation was analytical error.

It is not suggested that this occurs every time, because it does not, and examples could be quoted from Crawford's paper to show that triplicate determinations have made no difference.

It has been suggested above that reliance should not be placed on a single determination. This is primarily to guard against that occasional big deviation, because, if a big difference occurs, a third determination can be made.

Actually, the greater degree of accuracy obtained by repeating determinations is given in the following, assuming a probable error of 0.065:

Number of determinations within ± 0.1 = 70 per cent on singles
= 86 per cent on doubles
= 92 per cent on trebles

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DISCUSSION

(*G. B. Gould presiding*)

H. F. Hebley,* Pittsburgh, Pa.—Dr. Grumell mentions a misconception encountered in England, where many individuals interested in coal interpreted the tolerance of plus or minus one unit to mean that on a 10 per cent average ash coal the reading would be either 9 or 11 per cent. With such an erroneous idea, criticism that the allowable tolerance was too wide was to be expected. I have never encountered this particular misconception, but we have many of our own, one of the most usual being the following: Given two samplers, A and B, each collecting an independent sample from the same lot, no matter whether that lot is a carload or a consignment of many carloads; if the determination reported on A's sample is 10 per cent ash, and the determination on B's sample is above 11 or below 9 per cent, B did not obtain a representative sample. This interpretation of the new American Society for Testing Materials Commercial Procedure for sampling coal is a prevalent one.

The stress laid on the influence of the number of increments as it affects accuracy is most timely. In studying the sampling methods carried out by the coal industry, it is rarely found that the requisite number of increments is taken in actual practice. Possibly in many cases this is caused by the heavy flow of coal, which precludes any practical method of avoiding gathering an increment of unwieldy proportions. At some coal-preparation plants one increment of approximately 60 lb. is all that is taken. It has been our experience that almost as much difficulty is encountered in understanding and interpreting the literature and specifications as in finding coal men that are willing to cooperate in supplying the means—money, men, and equipment—to carry out the recommendations provided by the specifications.

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There seems to be a definite need for clarification of the words "sample," "observation," "increment," etc. For instance, it is stated that within certain modifications the Average Error increases with the ash content of a coal. If there are enough observations of a certain characteristic of a coal, the Mean and the Average Error can be calculated. On what are those observations based?

In Publication No. 405, British Standards Institution, pages 15 to 19, inclusive, two populations are compared:

1. Groups of tests, or observations, each one of which was drawn from single carloads of a consignment of many cars.

2. Groups of increments, or observations, all drawn from a single carload.

In case 1, if the text is construed correctly, each car of coal was discharged into a hopper from which the coal was raised by a bucket elevator. At the discharge point of the buckets 10 increments, or cuts, each weighing approximately 5 lb., were taken uniformly spaced in regard to time during the elevation period. Assuming that these 10 increments were mixed together to form one gross sample of 50 lb., is not this procedure averaging mechanically 10 unknown observations? Therefore, strictly speaking, should not the Probable Error be divided by the Square Root of 10, or the Square Root of 10 minus 1?

In case 2, dealing with a single carload, 10 increments, each of approximately 5-lb. weight, were again taken, but were kept separate and analyzed separately, yielding 10 observations. It is stated that "the Average Error for increments taken from a single carload has been shown to be *of the same order* as the Average Error for carloads in a consignment." It would be a great help to many in the industry if the phrase "of the same order" were amplified.

As brought out by E. S. Pearson in his paper before the Royal Statistical Society, some attention should be paid to the difference in terminology. To quote Pearson: "In the first place in this (Pearson's) paper, the term 'sample' is used in the statistical as distinct from the ordinary commercial sense. Thus, a sample consists of a number of individual units for each of which one or more characters have been measured and recorded; the measured values are often termed 'observations,' and it is also common to speak of the observations as forming the sample. Thus a small sample may consist of 2, 5, 10, etc., units, or observations, and a large sample of perhaps 50, 100, 500, etc."

In our own work, we are following the procedure that has been mentioned in Dr. Grumell's paper; namely, that of reducing the gross sample to minus 4 mesh before any reduction in quantity is carried out.

Dr. Grumell's further suggestion that a larger amount of material be taken when the sample is reduced to 8 mesh is a point worthy of serious consideration.

I note that the question of level of control is still unsettled. The British Standards Institution has set up a 1 per cent level (99 times out of 100 within plus or minus one unit of ash content). Such a measure calls for rather rigid work in coals of high ash content. This has been recognized by the British Standards Institution's specifications for the sampling of run-of-mine and large coal, wherein a tolerance of 1.4 units of ash is allowed on coal that contains over 15 per cent ash.

Personally, I like the 5 per cent level of control (95 times out of 100 within plus or minus $\frac{1}{10}$ of the average ash content), adopted by the A.S.T.M. Commercial Procedure specifications on sampling coal. Such a measure allows a gradual increase in tolerance as the ash content rises; and the level of control is equivalent to 2 times the Standard Deviation (3 times the Probable Error), making the necessary calculations more convenient. Also, in statistical work there is a growing tendency to select the 5 per cent level of control (2 times the Standard Deviation) as the point of significance in indicating whether the fluctuations encountered in sampling have been brought about by chance causes. Any deviation greater than 2 Standard Deviations must be considered as possibly being brought about by an assignable cause.

Deviations in excess of a 1 per cent level of control (99 out of 100) are highly significant of an assignable cause.

Of course, there is an economic limit for expenditures for control studies, dependent on the value of the commodity being studied, and with coal it may easily be uneconomical to make such expenditure to conduct frequent search for the assignable causes when 2 Standard Deviations are used as the boundaries. Possibly 3 Standard Deviations would be more suitable in some cases.

The foregoing remarks apply directly to the control chart so briefly touched upon by Dr. Grumell. It is to be regretted that he could not see his way clear to amplify this important phase of statistical application.

The necessity for the adoption of modern methods of sampling for industrial work is very urgent. All boiler test work is predicated on the accuracy of the determinations yielded by the sample of coal taken during the test. In the majority of cases the sample collected leaves much to be desired. Considerable pains will be expended to obtain comparatively accurate readings from various instruments, such as thermometers, pyrometers, draft gauges, Orsat apparatus, etc. Observations will be taken every quarter of an hour for a 24-hr. period; yet the calorific value, which yields the denominator in the calculation for efficiency, is often open to question, owing to poor sampling.

In a specific case, five different calorific determinations of the same coal were available. This coal was being used on an acceptance test for a modern steam generator. The American Society of Mechanical Engineers indicates that any efficiency within plus or minus 3 per cent of the guaranteed efficiency should be considered a substantial compliance with the guaranteed figures. However, such an understanding must be made beforehand between the client and the manufacturer. If such precaution is not taken, the boiler efficiency must meet the guarantee. In the case under discussion there were five analyses, each of which could have been used on the test. Some of these determinations, had they been used, would have indicated that the boiler did not meet its guarantee.

Dr. Grumell touched upon the question of moisture present in the coal. The determination of moisture content of coal, especially on samples collected during a boiler test, is, at the present time, unsatisfactory. Many methods have been suggested, but they all suffer from the criticism of inconvenience or inaccuracy. At the present time we collect a duplicate sample especially assigned to the determination of moisture. Periodic cuts, or increments, are immediately placed in a standard 5-gal. milk can, and the lid quickly replaced to prevent moisture loss. Upon delivery to the laboratory the air dry loss is determined, and this moisture determination is compared with the moisture determination yielded by the sample being used for the derivation of the proximate analysis.

Routine test work is also most important, and, unfortunately, a great deal of this routine work is being carried out on samples that are inadequately collected and prepared.

The growing tendency toward the adoption of 1.6 sp. gr. as the division point between clean coal and refuse should receive consideration. Morrow and Procter, and Bushell, have used this line of demarcation to advantage. I agree with the tolerance of 0.2 and 0.3 in the Average Error, and because the reduction of the gross sample may be a very great source of error, I tend to agree with the suggestion that coning and quartering be abandoned, and that crushing and riffling be substituted, as I feel that the errors of sampling, reduction, and analysis may easily outweigh the variability of the coal itself.

The time and labor that various interested individuals throughout the world have contributed to this subject are gradually becoming appreciated by the coal industry; and it is safe to say that ere long the necessary mode of thought required in viewing

results obtained from coal sampling will be commonplace to a goodly percentage of those associated with this industry.

When it is fully appreciated that a tolerance should be adopted in the measure of a coal's value, the coal industry will have taken a distinct step forward.

D. R. MITCHELL,* State College, Pa.—Dr. Grumell has made a splendid summation of the progress in coal sampling during the past 10 years. There is little that can be added.

The charts in Fig. 2 should be further emphasized. Often it is assumed at a given coal mine after a cleaning plant is installed that a uniform product will be made during the life of the plant. This is not necessarily true. Not only may the level of control change from year to year but coal from different sections of the same mine may show widely differing washability characteristics. If coal from such mines is not thoroughly mixed in definite amounts prior to entering the preparation plant, plant operation is made difficult, the finished product may vary widely in analysis, and trouble may be caused in consumer plants. Or coal from a section of a mine differing in washability and combustion characteristics from the coal in other sections

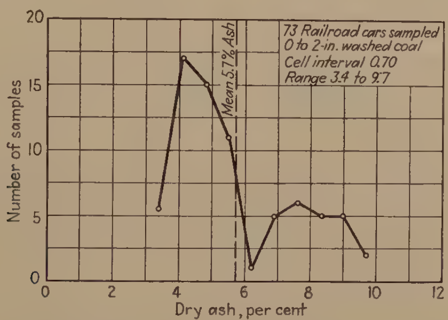

FIG. 6.—FREQUENCY POLYGON OF COAL WITH BIMODAL CHARACTERISTICS (TWO PEAKS).

of the mine can be segregated, cleaned on a night shift and shipped if necessary to a selected list of consumers.

Several engineers are attempting to use the theory of small samples in predicting the range in quality of future coal shipments. However, care must be used on account of the condition likely to be encountered as described in the preceding paragraph. Most coals follow the "normal" curve; occasionally one is encountered showing bimodal characteristics—two peaks instead of the one peak of a normal frequency distribution. Fig. 6 shows this clearly, the data for which were obtained on a coal cleaned in a modern plant. These data fall into two distinct frequency distributions and it is obvious that a small number of samples might all be within one or the other of these distributions. Under such conditions it would be impossible to predict the probable range or frequency distribution of shipments of this coal from a small number of samples.

R. M. HARDGROVE,† New York, N. Y.—We agree entirely with Dr. Grumell that the size-weight-ratio is the logical starting point for coal sampling and when coupled with Bailey's riffle tests, which established the effect of free ash on the accuracy of sampling, we really have a scientific method of attacking this problem.

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† The Babcock and Wilcox Co.

Free ash is the real variable rather than total ash, but as Dr. Grumell points out it can be estimated from a general knowledge of preparation methods and types of coal and can be checked by float and sink methods.

We cannot agree that a level of accuracy of 1 per cent once in 100 is ample. Surely for accurate work, as in boiler tests, an accuracy of 1 per cent once in 10,000 is justified. It does not always require a very large sample to obtain this accuracy, as better cleaning methods have reduced the free ash percentage and the smaller sizes more frequently used both tend to reduce the size of gross sample required.

Table 19 gives the sizes of gross sample required for three standards of accuracy and varying percentages of free ash.

TABLE 19.—*Weight of Gross Sample Required for 1 Per Cent Deviation*

Size	Free Ash, Per Cent............ Weight Ratio..................	5 0.025	4 0.032	3 0.045	2 0.076	1 0.265
1 in 10,000	2-in. maximum size.......	1,600	1,250	890	525	150
	1¼-in. maximum size.....	800	625	445	262	75
	⅝-in. maximum size......	120	94	67	40	11
1 in 1,000	2-in. maximum size.......	1,180	920	660	388	110
	1¼-in. maximum size.....	590	460	330	194	55
	⅝-in. maximum size......	89	70	49	30	8
1 in 100	2-in. maximum size.......	675	526	375	222	63
	1¼-in. maximum size.....	337	263	187	111	31
	⅝-in. maximum size......	67	40	28	17	5

Sizes are based on round-hole screens.

Maximum size of impurity estimated to be 2 in., 0.40 lb.; 1¼ in., 0.20 lb.; ⅝ in., 0.03 lb.

Influence of Mechanization on Location of Coal Production in Illinois

By Paul Weir,* Member A.I.M.E.

(Chicago Meeting, October 1938)

During the past decade, methods of producing bituminous coal in the state of Illinois, which ranks third in production among the states in which bituminous coal is mined, have undergone great changes. From less than one million tons in 1923, the strip-mine production has gradually increased to a high of 11,725,870 tons in 1937. From a negligible amount prior to 1927, the amount of coal mechanically loaded underground had increased to a high of 28,374,362 tons in 1937. The amount manually loaded in 1937 reached a low of 12,332,023 tons. In this same year, this state ranked first in strip-mine production, also in mechanically loaded production. For this reason a study of the influence of mechanization on location of production in Illinois may be helpful in anticipating what may happen in other producing states.

The reason for the changes in methods lies in the differences in costs of production. The Bureau of Research and Statistics of the National Bituminous Coal Commission has made public the following costs of production by various methods in the state of Illinois (district 10) for the last nine months of 1937: all mines, $1.7602 per ton; strip mines, $1.4319; mechanical loading mines, $1.7457; hand loading mines, $2.1793; all underground mines, $1.8703. These are weighted averages for the entire district.

Inasmuch as production in 1937 was reported from 58 counties scattered over two-thirds of the area of the state and from five different seams varying in thickness from less than 3 ft. to as much as 12 ft., it is apparent that the location of production must be influenced by the extent of mechanization possible in the various producing counties.

Shift in Production

The term "mechanization" is commonly used to designate the coal-loading operation. However, in a broader sense it embraces the use of any and all mechanical and electrical equipment on the surface and underground. Strip mining is an intensive form of mechanization. The mechanical cleaning of coal is a form of mechanization. Cutting machines and motor haulage likewise are forms of mechanization, hence

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it may be said that mechanization was proceeding many years before the introduction of loading machines.

Basically, production in any area diminishes or ceases when operations become unprofitable or when reserves of minable coal are depleted. Operations become unprofitable when the realization from sales is less than the cost of production. When the proportion of manual loading in any area declines to any substantial extent, average sales prices become based to a large extent on cost of production obtained in mines employing mechanical loading and in strip mines. If natural conditions do not permit mechanization or strip mining, production shifts to another area.

Factors other than mechanization and depletion of reserves may accelerate or retard such a shift. Some of these factors are wage-scale differentials, freight-rate differentials, truck transportation and consumer requirements.

While the reserves in a particular community within a district may be depleted, according to estimates of the State Geological Survey for Illinois, only a very small percentage of the coal reserves in any individual district has been mined. Depletion, therefore, may be disregarded in so far as it may be a factor in shift of production from one district to another, except that a substantial shift may follow depletion of strippable areas.

For many years wage scales and working conditions in Illinois have been established by collective bargaining. Differentials between areas within the state have been maintained. With a fixed wage scale, the cost of production has been largely a product of natural conditions and methods. Where natural conditions have been favorable, operations have to a large extent been mechanized, with a resulting lower cost of production.

Freight-rate differentials among the various districts on coal moving into common markets have not been disturbed to any marked extent during the past 10 or 12 years, hence have not been responsible for any major shift of production within the state. Truck transportation probably has been responsible for some minor shifts, chiefly intrastate.

Consumer requirements have changed to some extent by reason of changes in burning equipment to permit the use of cheaper grades of coal. As with truck transportation, this factor may be responsible for minor shifts.

Significant shifts from one district to another have come largely through changes in cost of production resulting from mechanization and from the beneficiation of the coal as shipped by mechanical cleaning.

Districts Studied

For the purpose of this study the state has been divided into six districts (Fig. 1), which follow rather closely the freight-rate groupings.

The boundaries of the districts also are outlined so that within each district the inherent quality of the coal and the thicknesses of the principal beds approach uniformity. Each district then represents to a large extent a separate geographical and geological unit, which contains common problems within itself.

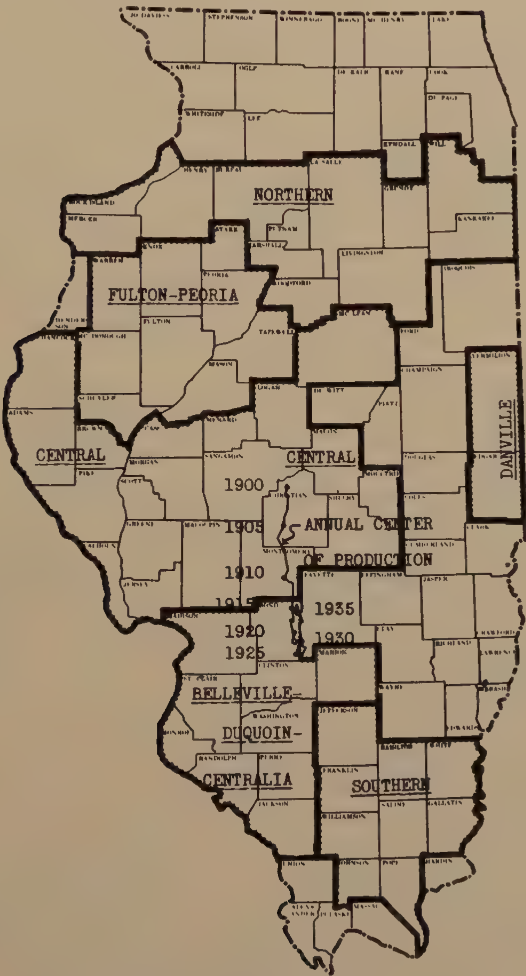


FIG. 1.—DISTRICTS STUDIED.

Table 1 shows for each district the principal beds being mined, together with the percentage of the 1937 production from each and their approximate thicknesses.

The charts in Fig. 2 show for each of the six districts the percentage of the state's production mined in that district by years. They also show the percentage of each district's production that came from strip

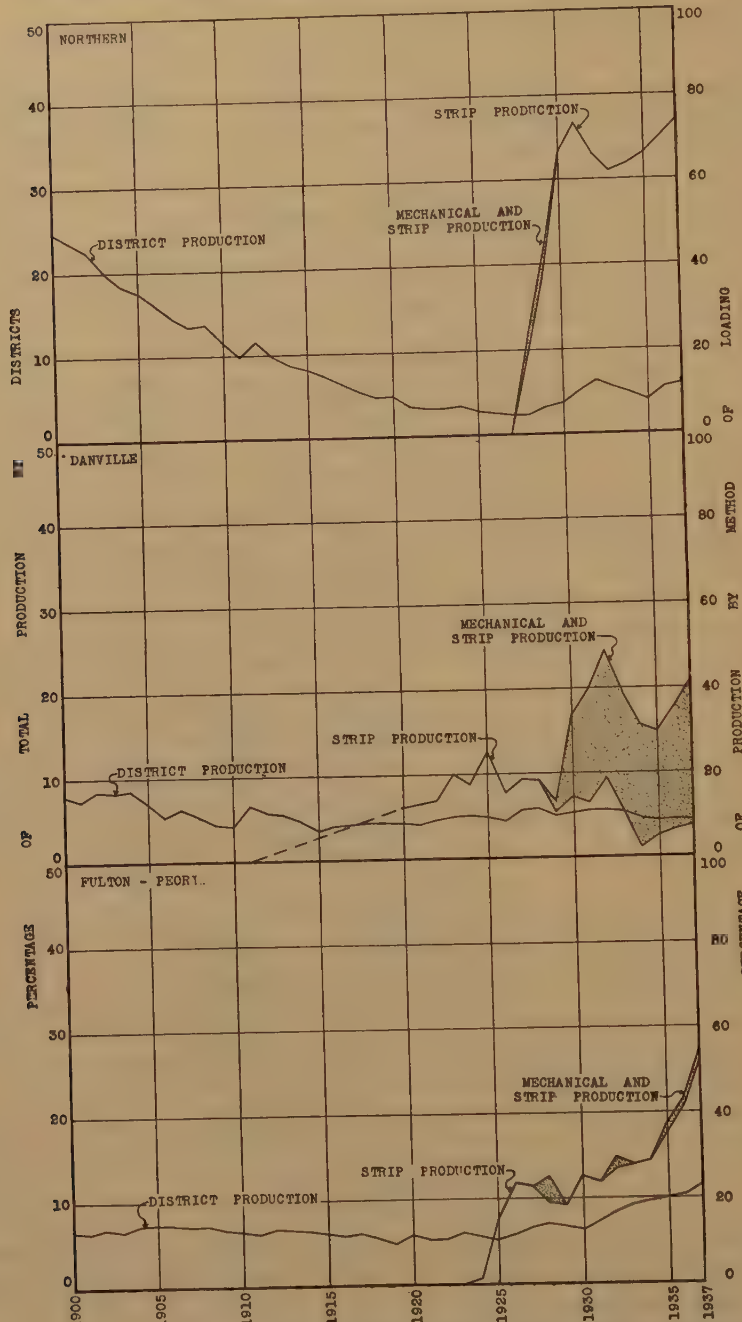
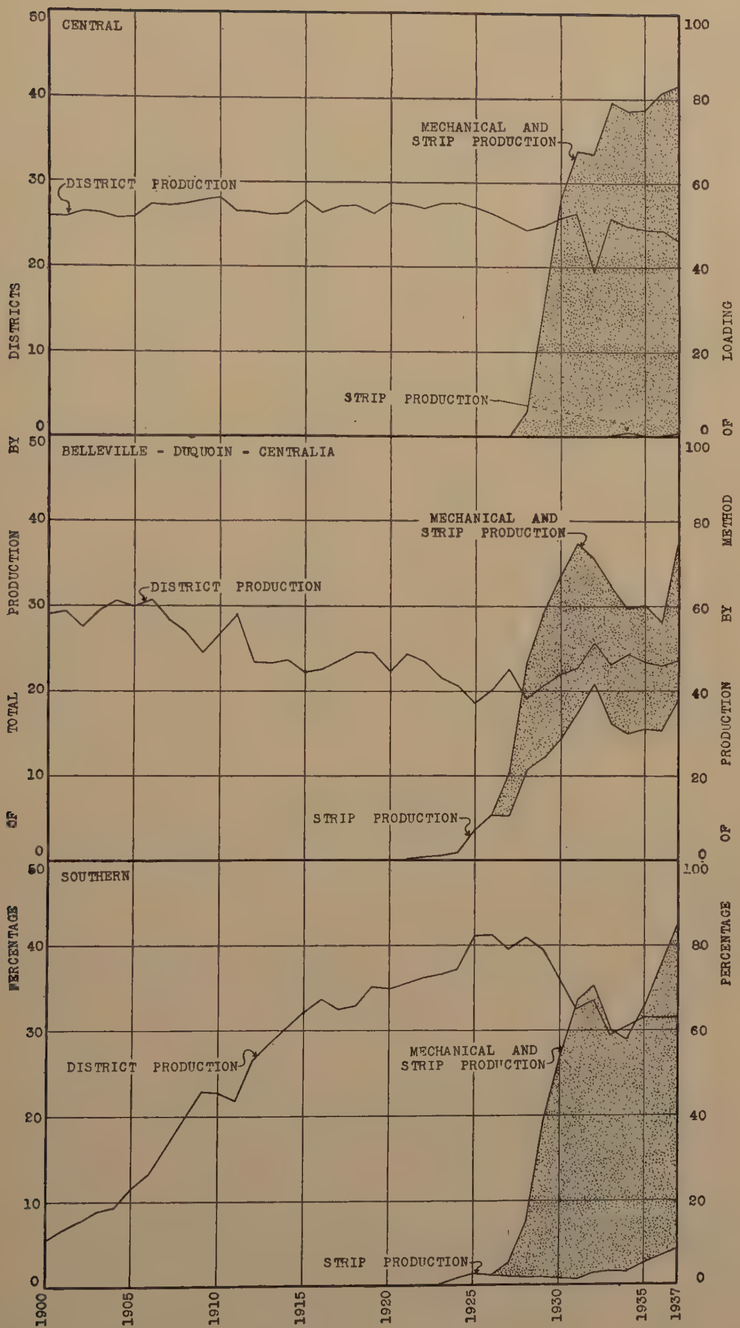


Fig. 2.—Percentage of coal production in Illinois mined



IN EACH DISTRICT, AND PERCENTAGE PRODUCED MECHANICALLY.

mines and the combined percentage of strip-mine and mechanically loaded coal.

TABLE 1.—*Principal Producing Coal Beds, Illinois*

District	Seam	District Production in 1937, Per Cent	Thickness
Northern	No. 1	3	4′8″
	No. 2	97	2′0″–3′0″
Danville	No. 6	91	6′0″
	No. 7	9	5′6″
Fulton-Peoria	No. 5	76	4′6″
	No. 6	24[a]	3′6″
Central	No. 5	20	5′8″
	No. 6	80	7′6″
Belleville	No. 6	100	7′0″
Southern	No. 5	20	5′6″
	No. 6	80	8′0″

[a] Includes approximately 4 per cent from No. 1 seam.

From 1900 to 1925 there was a gradual shift in production from the northern part of the state to the southern part. The lower cost of production arising out of the superior natural conditions of mining in the south more than offset the freight-rate differentials to common consuming markets in the north. These superior natural conditions permitted greater use of mechanical and electrical devices. Artificial conditions during the war years and afterward affected location of production. By 1925, these artificial conditions were largely removed and location of production tended to seek levels based on quality of coal, cost of production, and freight rates to consuming areas. At that time strip mining was becoming a factor in the state as a whole. In 1927, mechanical loading commenced and within a few years had grown to substantial proportions. The effect of mechanical loading and strip mining on location of production, then, is confined largely to the period subsequent to 1925.

A large proportion of the production of the Northern, Fulton-Peoria, and Belleville-DuQuoin-Centralia districts comes from strip mines. Strippable reserves in these districts are substantial. Strip production in the Danville district is declining because of approaching exhaustion of what are now considered to be strippable coal lands.

Because of the thin seams in the Northern district, there has been very little underground production loaded mechanically. While the seams in the Fulton-Peoria district are somewhat thicker, conditions of roof and bottom are not very favorable to mechanical loading underground. In addition to this, this district probably contains the largest

amount of strippable reserves to be found in any district. Very few strippable areas are found in the Central district, hence there is little production from this source. A minor amount of the Southern district's production comes from strip mines.

With the known lower costs of production by strip mining, a substantial shift to districts in which strip mining represents a substantial proportion of the production would be expected. For this reason Northern, Fulton-Peoria, and Belleville-DuQuoin-Centralia districts have gained some production at the expense of other districts. Mechanical loading has probably had only a minor influence on interdistrict shifts. The underground mines that remained in operation on a hand-loading basis when production was seeking a level in 1925 were those in which natural conditions were superior and which therefore, except those in the Northern and Fulton-Peoria districts, could be fully mechanized. The intradistrict shifts have probably been of greater moment than the interdistrict ones. Mechanical loading has prevented shifts to a greater degree than it has caused them.

On Fig. 1 is shown the location of the center of production of the state as a whole for each five-year period commencing with 1900. The center of production during the first 20 years of the present century was constantly moving southward. Since 1925 the general trend has been north, which reflects the growth of stripping in the Northern and Fulton-Peoria districts.

Various Conditions Affect Shift

Artificial conditions in other producing states during and subsequent to the World War parallelled those in Illinois. By 1925, the percentage of the nation's production mined in Illinois had sought its level (Fig. 3). This chart also shows the growth by years of strip mining and mechanical loading in the state and indicates the relationship between the growth of strip mining and mechanical loading and the level of production in the state.

The quality of Illinois coals varies among districts, hence the coals are not strictly interchangeable from a use standpoint. However, when differences in costs of production among districts are approximately constant, there is a minimum of shift from one district to another because of differences in quality of coals. Under a fixed set of conditions, location of production is more or less stable. When, because of mechanization, the level of production costs in any district affects existing differences in production costs, there is some shifting among districts regardless of quality.

Mechanical cleaning is exerting an ever increasing influence on location of production. The degree of beneficiation possible on inferior coals is greater than on the better coals. The tendency of this is to

make the quality of mechanically cleaned lower grades approach the quality of the raw higher grades. No statistics are available to indicate this trend. However, at present the installed capacity of mechanical cleaning plants is approximately as shown in Table 2. The highest

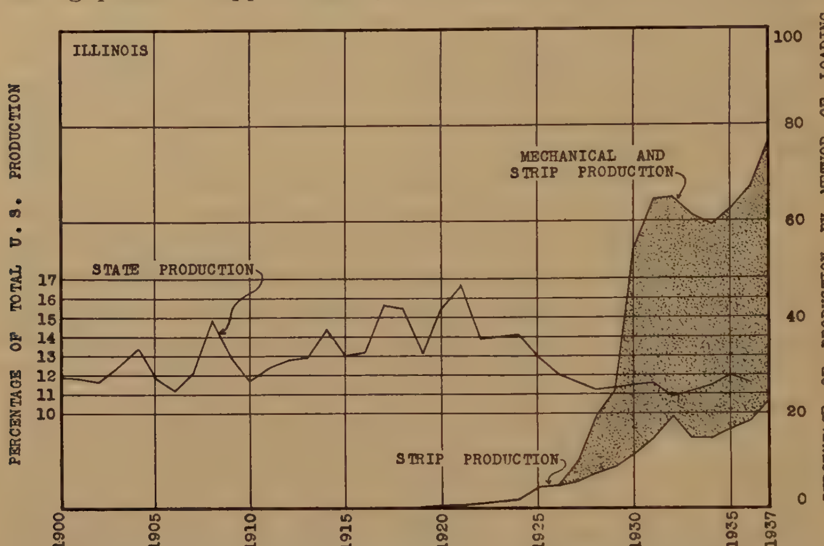


FIG. 3.—PERCENTAGE OF NATION'S COAL PRODUCTION MINED IN ILLINOIS, AND GROWTH OF STRIP MINING AND MECHANICAL LOADING IN THE STATE.

grade coal in the state is generally considered to be that mined in the Southern district. The probable effect of cleaning in other districts, then, is to shift some production from Southern to those districts.

TABLE 2.—*Installed Capacity of Mechanical Cleaning Plants, Illinois*

District	Number of Plants	Approximate Annual Production, Tons
Northern	2	1,300,000
Danville	1	450,000[a]
Fulton-Peoria	5	2,500,000
Central	2	450,000
Belleville-DuQuoin-Centralia	12	3,100,000
Southern	9	3,700,000
Total	31	11,500,000

[a] Under construction.

Despite the tremendous increase in mechanization in Illinois, with the resulting decrease in manual loading, there have been no precipitous shifts in production from one district to another. While the shift may have been acute in so far as individual companies in a district are concerned, in the state as a whole Fulton-Peoria is the only district in which

production is now at the highest level since the beginning of the present century. This shift to Fulton-Peoria is the result of strip mining, with its low cost of production, together with a high degree of beneficiation of the coal by mechanical cleaning. Also, this district has an inherent advantage of geographical location.

Future developments in mechanization in the districts that have an advantageous geographic location (relatively lower freight rates to common consuming markets) will undoubtedly result in additional shifts. When all mines are on a hand-loading basis, freight-rate differentials among districts to common consuming markets may be more than offset by lower costs of production resulting from advantageous natural conditions of mining existing in the district having the highest rate. However, mechanization, including mechanical cleaning, in districts having inferior natural conditions of mining but an advantageous geographic location, may lead to a cost of production and quality of coal that will make delivered sales prices competitive with coals from districts having superior natural conditions of mining and better quality of coal as it occurs in the seam but a higher freight rate to common consuming markets.

The statistics on which the data shown on the charts in Figs. 2 and 3 are based were taken from the Annual Coal Reports of the Department of Mines and Minerals, State of Illinois.

DISCUSSION

(*John A. Garcia, Jr. presiding*)

W. H. Voskuil,* Urbana, Ill.—Mr. Weir's paper, in one respect, may be regarded as a case study in the effect of technological improvements upon the cost of producing coal. With the relationship of wage scales and transportation rates among the several mining districts remaining relatively constant, the market opportunity of a producing district or individual mine is widened to the extent that costs, other than wage rates and freight tariffs, can be reduced. In Illinois this is accomplished to a considerable extent by strip mining methods and mechanical loading. Both have made rapid strides in this state and each accounts for one-third of the coal stripped or mechanically loaded in the United States in 1937.

The significance of these technological developments in the Illinois coal industry go beyond the regional shifts of coal production as portrayed in Mr. Weir's paper. They affect also the competitive position of coal in the general fuel market. With coal, oil, gas, and, to a limited extent, water power, competing keenly for the energy market, each must take advantage of whatever weapons it possesses to stay in the market or to gain a larger share of it. For coal, this weapon is, in general, a lower cost per unit of heating value than the cost of competing fuels. With this weapon coal must seek to overcome some of the obvious advantages possessed by the liquid and gaseous fuels. The development and further extension of mechanized mining, either by strip-mine methods or mechanical loading in shaft mines, appears to be the most effective means available to the Illinois coal industry to maintain or improve its competitive position in the general energy market. This advantage is aided and supported by the construction of coal-cleaning plants, thereby opening markets to inferior coals that may be near large consuming markets which were not open to them until cleaning processes for the improvement of the coals became available.

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Mechanical Mining by the Consolidated Coal Company

By G. Stuart Jenkins,* Member A.I.M.E.

(Chicago Meeting, October 1938)

Conditions at the properties of the Consolidated Coal Co. had reached a point where improvements were almost impractical. The mines, sunk years ago, had shafts and entries so small as to preclude the use of large mine cars. A radical change was necessary in the type of mining and also in the type of mine to accommodate such haulage units.

After due consideration, the company decided to enter the field of trackless mining, utilizing belt conveyors and chain-flight conveyors, the belt conveyors to take the long haul, and the chain-flight conveyors to be used for gathering. It was decided not to sink a shaft, but to drive a slope on an inclination of 17½°, which the company believes to be about the limit on which coal can be carried on a belt without rolling back unduly.

Sinking the Slope

It was decided, in sinking the slope, to make an opencut until rock was reached, which was some 65 ft. below the surface, and then to drive the slope through an inclined tunnel from that point. In making the opencut, the schedule called for the removal of about 25,000 cu. yd. of clay, holding the banks on an inclination of one and a half to one, the thought being to make this cut in a minimum of time, put in a reinforced concrete slope, and then back-fill. Difficulty was experienced because the clay was of a sort that was called "subsiding base" material by contractors visiting the job. In other words, a "squeeze" occurred, the pressure causing the floor of the cut to move up just as it does in many mines where the coal is underlain by thick fire clay and inadequate pillars have been left. It was necessary to move the waste banks back from the edge of the cut 700 or 800 ft., and even then the floor could not be controlled, the result being that almost 100,000 cu. yd. of clay was handled before the rock was reached.

A concrete slope was then rapidly constructed, using 95 tons of special-shaped reinforcing iron. The floor of this slope was made 30 in. thick at the bottom, and 17 in. thick at the top, the sidewalls being 18 in.

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* General Superintendent in Charge of Operations, Consolidated Coal Co., St. Louis, Mo.

thick at the bottom and 15 in. thick at the top. The center wall was carried 12 in. thick throughout. This entire structure was poured in about two weeks, and back-filling was begun at once. Even then, it was difficult to hold the structure in place, because of the heaving action of the clay.

In due time, a two-compartment slope with back-filling was finished, one compartment being 8½ ft. wide by 7 ft. high; the other, 6 ft. wide and 7 ft. high with 4-ft. openings in the center wall every 20 ft. to provide a manway, and to make easy inspection of the belt compartment from the material compartment.

In driving the slope through the rock, a radical departure was made from the usual methods. A belt conveyor was installed to haul the waste material out of the slope and to dump it onto a pile. It was removed with a 7-yd. scoop pulled by a Diesel tractor, which loaded the material, hauled it to the waste pile, and dumped it, all movements being controlled hydraulically by the tractor driver.

The rock tunnel was driven 16 ft. wide and 7 ft. high, the rock being drilled by pneumatic jackhammers, using patent drill bits to overcome the difficulty of sharpening bit steel. The work was done on a three-shift basis, using four men and a boss on each shift. Two jackhammers were used at a time with two men at each hammer. The rock was then shot down and loaded with a loading machine onto a flight conveyor, which in turn emptied onto a belt conveyor. All rock was blasted with a gelatin permissible, using electric detonators, both instantaneous and up to fourth delay. For obvious reasons, it was necessary to use a flight conveyor at the face of the slope. This flight conveyor was extended in 6-ft. sections until a length of about 150 ft. was reached, then the belt conveyor was extended and the conveyor was shortened and moved ahead. The same routine was repeated. On this three-shift basis it was possible to average 22 ft. per day for the entire tunnel, although some days 30 to 33 ft. was driven in 24 hr. However, these high records were lowered in the average by the time consumed in extending conveyors. The slope was driven down through the sixth seam into the fifth seam, and the latter was mined for several hundred feet, turning off four rooms, two on each side. The slope was then stopped, and this fifth-seam extraction utilized as a sump, thus providing dry work in the coal seam for future developments.

The Main Entry

Next, the main entry was started, turning off on a 60-ft. radius curve from the material compartment of the shaft, and running at about an angle of 60° from the direction of the slope. This first entry was to be utilized in the final setup, as a material track, and therefore a 60-ft. vertical curve was used to connect the slope to the coal seam. The entry

was driven 100 or 200 ft., then crosscuts were turned to pick up the haulage entries and the air-course entry, which in turn were driven back so that they were immediately over the portion of the slope that went down below No. 6 seam. All this development was driven with 10-ft. entries and 10-ft. crosscuts, the entries being on 60-ft. centers, with crosscuts every 60 ft. Ventilation was established temporarily by bratticing the manways or openings between the two compartments of the slope, and providing a fan on the material side of the slope, which forced the air down one side through the development and out on the conveyor side of the slope.

The plan of operation called for a hopper over the belt conveyor, so that a suitable reciprocating feeder could place the coal uniformly on the hoist belt. This hopper was driven from the bottom upward; the method of procedure being to utilize the temporary belt conveyor, with which the slope had been sunk, using a chain-flight conveyor at the loading end. A 4-ft. hole was driven upward into the second entry, which was to be used for haulage. Instead of scaffolding, the waste was allowed to fall in a pile at the bottom of the hole, and the men worked from this pile of waste material, keeping open only sufficient room for easy access. The hole having been driven through into the second entry, the waste pile was loaded by a mobile loading machine onto flight conveyors, which in turn loaded it onto the belt conveyor, which discharged it at the top of the slope.

The next step was to build a hopper. Suitable supports were put in place, and the hole was increased in size by enlarging the opening at the top, allowing the waste to fall down through the hole into a chute, which, in turn, fed it onto the belt conveyor, which at this time had been extended. In this manner, the rock from the hopper rolled down the hole without the expenditure of any manual labor. This first hopper will hold about 100 tons. A second hopper of about 300 tons capacity was planned for the third entry, this being possible because the slope was 45 ft. below the coal at that point, whereas it was only about 25 ft. below the coal at the second entry.

Suitable ventilation was then provided, so that the brattices in the slope could be removed; the whole shaft was used as an air return. As it was decided to make this a conveyor mine, the question arose whether to use conveyors to the exclusion of all other types of transportation, and it was decided that the first step would be to install in the main entry mother belts extensible to 2000 ft., discharging into large cars on the main entry, these large cars being used in what might be termed "shuttle service," and being of drop-bottom design, 21 ft. long, 7 ft. wide, 54 in. high, to carry 10 tons of coal when filled at water level. The general construction of the car provides for the installation of 10-in. side boards, increasing the capacity of the car to 15 tons.

Panel Entries

In driving the panel entries, the first step is to use drag flight conveyors, which consist of a driving head and a tail piece with pan sections 6 ft. long, which can be inserted between them as needed. The chain section with flights is 12 ft. long, taking care of both the upper and the lower run. One of these driving heads is elevated to discharge into the car. A three-entry system is used with a driving head in each of the other two entries. Coal is taken from the other two driving heads onto the first one by means of a cross conveyor of the drag flight type.

The flight conveyors used in the entries are similar to those later to be used in the rooms, and are of 20 hp., 15 in. wide, and extensible to a length of 300 ft. No difficulties have been experienced in carrying a full conveyor load for this distance. The conveyor can be extended to 400 ft., but, when of that length, some difficulty is experienced as the chain tends to have a jerky motion. The entries are extended until the conveyors are 300 ft. long, then the conveyors are shortened and moved up, a belt conveyor taking the place formerly occupied by the drag flight conveyor. The belt conveyor empties into the shuttle car.

The mother belt being installed in the center entry and suitable crosscuts having been driven opposite the room necks, 20-hp. drag flight conveyors were installed, discharging onto the mother belt and extending through the crosscut from the center entry to the outside entry and on into the room neck. Room conveyors were installed until four consecutive rooms on each entry were equipped with conveyors, each discharging onto the mother belt. In this way, one loading machine in the three entries provides development, another loading machine has a territory on one side of the entry, and a third machine has a territory on the other side. Consequently, the output of three loading machines is discharged on one mother belt, which in turn drops the coal into the shuttle cars.

Room Centers

While the ideal room centers have not as yet been determined, the ideal spacing will be somewhere between 50 and 80 ft.; the checkerboard system will be used, modified to give as large an extraction as the top will permit. This will be somewhat greater than in earlier operations, because with conveyor mining the extraction is more rapidly performed.

Breaking Down the Coal

As may be easily realized, conveyor mining calls for on-shift shooting, which makes compulsory the use of Airdox or some similar means of breaking down the coal. The coal is cut with a shortwall cutting machine, and these machines must cross over the conveyor, if each of them is to

serve four rooms. To expedite this travel, a transfer truck on endless tracks is utilized. This truck has the usual tilting pan, and tramming is effected by two individually controlled motors, each driving one of the endless tracks, thereby permitting the transfer truck to be turned in the same manner as a tractor. The truck is furnished with power by a feed cable extending to the panel entry.

On entering a room, the truck trams to the point where the machine is to be sumped, and the machine is then operated as in the usual type of mining. After traversing the face, the truck is moved over to the machine at the opposite side of the place, and the machine loaded in the customary manner. To reach the next room, the machine, because of timbering or some other obstacle, may be unable to reach its destination without crossing the room conveyor. If so, several ties are thrown into the pan of the conveyor and an inclined ramp is placed on each side. The truck and machine then climb over these ramps and ties and on into the next place to be cut. Thus it is not necessary for the machine to go back into the panel entry, as it can pass through the first open crosscut back from the face.

Ordinary electric drills are used, but a rubber-tired pushcar, resembling a glorified banana wagon, replaces the typical flanged-wheel drill truck. This pushcart is moved easily by the drillers, who work in pairs. The customary Airdox system is used for breaking down coal, but the air is piped into the territory, so that the compressor does not have to be moved up to the face. To eliminate a pressure drop in discharging the Airdox container, remote-control discharge valves are provided, which operate satisfactorily.

Experiments are being made as another means of breaking down the coal with a hydraulic snubber, which consists of a rubber tube encased in a woven sheath, placed in the hole instead of the Airdox container. This tube is expanded by a pump, which consumes about 7½ hp. and delivers in about one minute a hydraulic pressure of about 2500 lb. per sq. in., which is transmitted from the pump to the tube by a flexible hose. This method has been on trial for about six months at this property, and has reached such development as to promise successful commercial use in the near future. It requires the same size of hole and about the same number of holes per face as Airdox. These holes, however, must be carefully spaced or the coal will not be broken sufficiently to allow the loading machine to lift it.

Moving the Coal

In delivering material a rubber-tired pushcart also is employed. This, however does not give entire satisfaction because the men can push only a limited load; therefore a small rubber-tired tractor is to be provided, which, in turn, will pull a rubber-tired semitrailer carrying the

material. This tractor will be driven by an electric motor attached to storage batteries. One tractor can pull three or four semitrailers. Unhooking one trailer, it will spot it in position for unloading and go back to place the other loaded trailers. It will return to pick up the empty trailers.

When this system is in operation, the loading machine will be in one room; the cutting machine will be cutting the room that the loading machine has just left, and the drillers and shooters will be preparing the coal in the room that has just been undercut. The fourth room will act merely as a reservoir, to equalize the varying differences in the time required for these several operations. Also, it will give opportunity for the extension of room conveyors, which, of course, will have to be provided daily, as it is evident that more than one cut will be made per room during the 7-hr. shift, two cuts being an advance of about 18 linear feet, which will permit the insertion of three pan sections in a flight conveyor. One, two, or even three-pan sections are added at a time.

Development and Production

It has been the policy of this company to extend development just ahead of production, and this practice will be continued with this type of mining. As soon as the panel entry has been driven far enough to provide four room necks, production starts and thereafter follows immediately after development. As much advance must be made by the development machines in the three 10-ft. entries as the production machines may require. All three must progress practically cut for cut.

The present thought is to run these panel entries 2000 ft., and then head them off, as experience has indicated that ½ mile of conveyor belt can be operated satisfactorily. As the development progresses, the mother belt is extended, usually every 300 ft., or one flight-conveyor length. Tests, however, have indicated that it may be more economical to extend the mother belt only once for every 600 ft. of advance and to utilize the service of one more drag flight conveyor to make up this difference.

In loading the shuttle of cars from the mother belt, an interval occurs when one car is filled and an empty one comes into place. To handle the coal during this interval, just a few seconds, a small hopper, or perhaps a suitable arched end overhanging the car may be used. Another system, which operates very satisfactorily, perhaps better than either of the two ways mentioned, is to have the man at the carloading point in charge of the start and stop buttons of the mother conveyor. Then, by interlocking electrically the mother conveyor and the drag flight conveyors, pressing the stop button will stop everything and pressing the start button will start the mother conveyor, after which the drag flight conveyors will start through time-delay control, a period of one or two seconds being sufficient to keep the peak load on starting from being excessive.

Belts

The main hoist belt at Buckhorn slope is unusual in the Middle West as it uses a dual drive; that is, two motors and two drive pulleys. One of these motors is 100 hp. and the other, 40 hp. These motors are placed on the return side of the belt about 75 ft. back from the head pulley and are of the slip-ring type. Control equipment provides for the 40-hp. motor to be limited to that rating, and the 100-hp. motor takes up the load differential. This is accomplished by the motor characteristics, and also by providing a fixed resistance in the secondary circuit of the 40-hp. motor. This design provides that the two pulleys can take their respective loads without belt slippage on the drive pulleys, which would occur if the two-drive pulleys were connected mechanically. This slippage would be the result of varying belt stretch as the belt tension is reduced in going around the first drive pulley.

Belt stretch in the complete conveyor is controlled by a weighted take-up pulley, which is placed above ground on the slack side of the drive pulleys. The weight required is that sufficient to counteract the pull of the belt from this point, down to the tail pulley.

In starting, the belt stretch is about 6 ft. In the running position, however, this stretch decreases to about 2 ft. when loaded, and to a few inches when running idle. In the layout, provision was made for stretch in the belt by supplying suitable take-ups at the tail pulley and allowing extra travel in the weighted take-up. However, to date this permanent stretch, about 18 in., is negligible, and occurred shortly after the belt was installed. The belt is 36 in. wide and is guaranteed to handle 300 tons per hour at a speed of 375 ft. per minute. Tests have shown that at that speed a load of 400 tons per hour may be carried without overloading the belt or causing slippage. The desirability of carrying large tonnages on narrow belts at high speed is being debated, but the Consolidated Coal Co. believes that this is the correct solution, even though, perhaps, the belt life may be shortened. In this way the belt will move its capacity in tons while the life is still in the rubber. The life of the belt is at least five years, and may be as much as ten years. After the 5-yr. period, deterioration in the rubber, perhaps, may cause some difficulty; however, that phase has not as yet been reached, and no trouble may be experienced at the expiration of the period.

Performance and Equipment Costs in Shaker-conveyor Mining of Anthracite Coal

By John S. Marshall,* Member A.I.M.E.

(New York Meeting, February 1940)

The purpose of this paper is to present to the profession data and experience obtained over a period of 5 years in the operation of 87 shaker-conveyor units, and the production of 2,169,638 tons of run-of-mine anthracite coal. The data are not generally available and should be of interest to those concerned with the operation of anthracite and bituminous mines. To simplify comparison, a ton of 2000 lb. of raw run-of-mine coal has been used as the production unit throughout this paper.

Mining Methods and Equipment

The mine is in the vicinity of Wilkes-Barre, Pa., in the Buried Valley area of the Susquehanna River. Seven veins were mined, varying in thickness from 18 in. to 8 ft. The cover overlying the veins varied from 200 to 800 ft. Three-fourths of the 1200 acres of property was covered with from 100 to 200 ft. of water-bearing gravels and clays, which prevented first mining under rock cover less than 50 ft. and pillar removal under rock cover less than 100 ft. The property was opened by two shafts sunk on the mountainside to avoid the sand and gravel present in the river valley. The veins dipped an average of 5° from the shafts. In certain areas the veins were practically flat and in some places a reverse pitch as high as 5° was encountered for short distances.

A daily output of 3800 tons was hoisted on a two-shift hoisting operation. The mining consisted of 75 per cent pillar removal, and of the total production in the later years of operation 80 per cent was loaded by shaker conveyors, chain conveyors and scraper loaders. Shaker conveyors accounted for 70 per cent of the total output; chain conveyors and scraper loaders for 10 per cent. All conveyor work was double-shifted and 25 per cent was triple-shifted. A total of 87 shaker-conveyor units were on the property, of which 75 were kept in operation at all times.

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Shaker Conveyors

Five different makes of shaker conveyors were used, all being light models (7 to 10 hp.) using light-weight troughing supported by ball frames. Both 250-volt direct current power and 440 alternating current

FIG. 1.—GENERAL ASSEMBLY OF A SHAKER CONVEYOR.

were used to operate the conveyors. Fig. 1 shows the type of conveyor generally used, together with names of the various parts of the conveyor unit.

Double-arm overhead drive units were used except where vein heights were 30 in. or less, when side drives were used to avoid cutting rock for height. Double-arm overhead drive units were found to be the most satisfactory type of drive unit even for low vein work where extra rock had to be taken to make room for this type of drive. Side-drive units, owing to concentration of stress on one side of the machine, caused excessive breakage and maintenance cost.

Several types of trough support were tried out and ball frames with universal tops were found to be the most satisfactory type. Ball-frame life obtained was about 2 yr. of double-shift service. After that length of time the frames were worn through. It is believed that manufacturers could considerably extend the life of ball frames by equipping the frames with a special alloy-steel wearing strip. It is also recommended that manufacturers experiment with balls made from various plastic materials or rubber composition, both from the standpoint of extending useful life of this equipment and reducing noise incident to operation of trough line.

Troughs with both riveted and welded construction were used and both types of fastening were found satisfactory. In order to prevent breakage of troughs, $\frac{3}{16}$-in. steel was substituted for the standard $\frac{1}{8}$-in. troughing. Although the steel troughs were not in service long enough for a final determination, it is believed that the heavier trough will have twice the life at 25 per cent higher initial cost than the lighter trough. Swivels were equipped with pendulum and jacks when more than five troughs were used ahead of the swivel.

The present design of swivel was never satisfactory. The practice of carrying the conveyor motion around a turn in the trough line through a hinged joint does not appear to be a practical principle. The strains set up in the trough line from the change in direction are very heavy and eventually destroy the troughs adjacent to the swivel, as well as the swivel itself. Pendulum-jacks are difficult to fasten securely enough so that the continual lashing of the trough line will not loosen them. It is believed that the manufacturers of this equipment should experiment with some other means of changing direction of the trough line. Possibly the trough connection at the turn should be broken entirely and a system of bell cranks and drive rods introduced to drive the conveyor beyond the turn.

Such a system has been used, but because of light construction and poor design has been satisfactory only for temporary operation of short distances (about 50 ft.). It is believed that the design and construction of bell cranks can be improved and the cost reduced to the point that, with the subsequent reduction in maintenance cost, the operator will install them to replace the 30° to 90° swivels now in use. As one of the most important features of the shaker conveyor is its ability in pillar work to follow crooked pillars, this feature should be improved upon

and perfected so that spillage and excessive trough breakage may be eliminated.

Drive units with 65 to 70 strokes per minute and length of stroke from 4 to 6 in. were most satisfactory under nearly all conditions. Higher trough speeds caused excessive trough breakage and lower speeds would not move the coal. Several gangway shakers of special type were purchased; while they worked successfully, reducing the cost of development work and speeding it up, their maintenance costs were high. These units were not in service for a sufficient length of time to definitely determine their value.

Motors and Power

Most of the electric motors for conveyor drives were second-hand shunt-wound 7 and 10-hp. direct current motors. Fair service was obtained from these motors but in a number of cases the motor windings were defective and motors ran considerably over normal speed, causing excessive trough breakage. The bearings of these motors were sleeve type lubricated by oil rings. This type of bearing did not prove satisfactory and there were a number of bearing failures. Some of the later units purchased were equipped with new type SK ball-bearing motors, which gave very satisfactory service. Shaker-conveyor motors should be completely protected with covers so that pieces of coal or rock cannot get into motor windings.

All of the conveyor equipment was started across the line. Fuse switches or De-ion circuit breakers were provided for motor protection. There were few if any motor failures due to starting across the line, as the units were all overhorsepowered and line drop of voltage provided sufficient starting protection. Most of the starting equipment commercially available at reasonable prices failed to stand up under mine service. A new type of starting control has been developed recently, consisting of two interlocking switches and a De-ion circuit breaker. The switches are so interconnected that the shunt-field coils of the motor must be energized before starting current is applied. This starting switch offers promise of fulfilling the need of a rugged type of starter at a reasonable cost. When overspeeding of motors occurred, resistance wire was introduced into the armature circuit to control the motor speed.

Further study should be made of variations in speed because of wide changes in direct current voltage under operating conditions existing in the mines. The writer is of the opinion that the use of small portable motor generators installed underground near the load centers should be adopted in mines largely equipped with shaker conveyors, replacing the general practice of a single large unit situated outside of the mine. Such small units could be moved easily from time to time as the load centers changed. Satisfactory operation of shaker conveyors on direct

current power cannot be obtained where there is a wide variation of voltage. Voltage under 220 will not give conveyor speeds high enough to move coal efficiently and continued operation on low voltage will eventually destroy the motor windings. Voltage over 275 causes serious overspeeding of conveyor motors and damage to drive unit and trough line.

Labor

Nearly all conveyor mining was done on a contract basis, separate car rates being paid for each vein. Conveyor miners paid their laborers on a car basis. This method of paying laborers was found to be more satisfactory than the general practice in the Anthracite region of paying laborers on a day rate, because of added incentive to each man on the crew. Cross timbers were paid for on a contract basis. Day rates were paid for setting up and tearing down conveyors, repairing conveyors, cleaning caves and for any vein conditions to which the contract vein rate was not applicable. The contract work was 75 per cent of the total work on conveyors. Some development work was done by shaker conveyor using a transporting conveyor on the gangway after its completion, to receive coal from the chamber and pillar units.

The number of men working on a conveyor unit depended upon the condition of the working face and varied from one miner and one laborer per shift to one miner and five laborers per shift. The average conveyor force consisted of one miner and two laborers. About 50 per cent of the conveyor mining was done in areas that were partly or totally caved or flushed, and about 15 per cent of the work was in veins 30 in. thick or less.

The conveyor miners furnished all tools required; 75 per cent of the units were equipped with either electric or compressed-air drills. All coal was blasted off the solid. The conveyor miners also lubricated all conveyor equipment and were held responsible for condition of machinery. This method of lubrication of equipment was not satisfactory and subsequent experience has proved that such lubrication should be taken care of by a man who has no other duties to perform. With the latest type of equipment, lubrication once a week is sufficient. One man can inspect and lubricate 10 units per day if they are not too widely separated. A sectional electrician who was responsible for the maintenance of all equipment in his area was employed for each mine section. His duties consisted of inspection and minor repairs to equipment, together with taking care of all other electrical work in the section. A machine foreman was employed whose principal duties were to check on installation and operation of all mining equipment and to follow up the performance of all units.

Mining Practice

The usual procedure in pillar mining was to advance up the pillar with a 6-ft. skip to the end of the pillar line and retreat with the remainder

of the pillar. Where old rooms were open, conveyor troughs were laid up the empty room and the entire pillar recovered on retreat. Owing to narrow pillars and the fact that in nearly all cases a skip had to be taken on the advance, the number of pillars that could be worked from one entry at one time was usually limited to three. Where attempts were made to work more than three pillars at one time, a squeeze and loss of

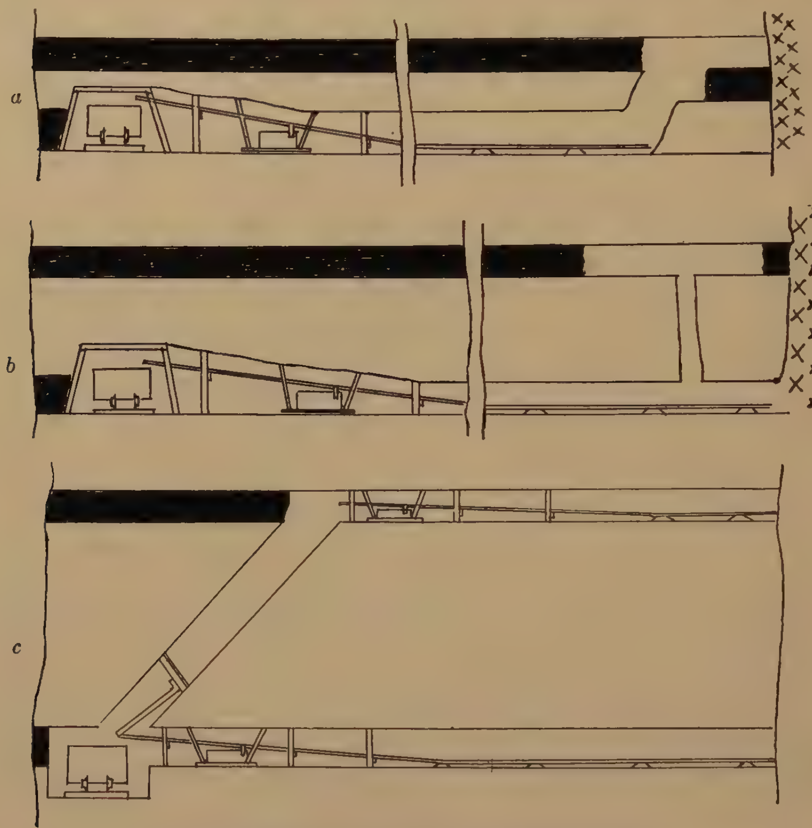

FIG. 2.—METHOD OF MINING CONTIGUOUS SEAMS.
a, rock interval 2 to 4 feet.
b, rock interval 4 to 10 feet.
c, rock interval 10 to 25 feet.

coal generally resulted; also, it became difficult to provide adequate transportation when more than three units were installed in one entry. The writer believes that the heavy thickness of water-bearing sands and gravels had a great deal to do with reducing the number of working places that could be maintained on each entry.

Conveyor mining was done in seven beds. Most of the conveyor work was carried out in the upper beds, reserving the thick lower bed for hand work. In most beds the best working places were reserved for

hand work and in the thin beds hand work was eliminated entirely except in opening conveyor places and extracting gangway pillars. Much of the work consisted of mining contiguous veins with a rock separation of from 1 to 25 ft. Such veins were mined by a combination of methods. When the rock thickness was not over 3 ft. the rock was taken with the veins and both veins mined as one. Where the rock thickness was from 2 to 4 ft., the lower bed was mined on the advance, the rock shot down and the upper bed recovered on the retreat, as shown in Fig. 2*a*.

Where the rock thickness was from 4 to 10 ft., the lower bed was mined on the advance and the upper bed recovered on retreat through small rock holes driven every 30 ft. from the lower to the upper bed. Coal from the upper bed was shoveled into these holes, which discharged coal onto the conveyor in the lower bed as shown in Fig. 2*b*.

When the rock interval was from 10 to 25 ft., the beds were worked separately. The development work was driven in the lower bed and the upper bed was mined through 45° rock holes as shown in Fig. 2*c*. In this type of mining, except where both veins were flushed, it was impossible to recover the pillars in both beds, as the removal of either bed destroyed the other. In such places about 75 per cent of the poorer bed was recovered and the remainder was left for support, while the better bed was removed as completely as possible.

All veins at the mine contained a large amount of refuse, the average bed section being 40 per cent refuse. Producers were required to gob as much of this material as possible and were disciplined for loading excessive refuse. Of the 40 per cent in the vein, over one-half was gobbed in the mine and the run-of-mine product produced 80 per cent clean coal. A standard 6-in. topping on the mine car was required. The conveyor miners were responsible for loading, topping and handling their cars at their loading chutes.

Conveyors were generally limited in length to 300 ft. but where conditions required were extended to 500 to 600 ft. by moving the drive unit into the middle of the trough line. Such long trough lines were expensive to maintain and resulted in high trough and drive-unit breakage. Pillars were generally narrow and often crooked, requiring the use of several swivels.

All conveyor drive units were held down by ratchet jacks, three or more jacks being required on each drive unit. An attempt was made to keep two connecting bolts in each trough section, each bolt equipped with two nuts and each trough section supported by a ball frame. Because of shortage of material, this was not always done.

The roof conditions encountered were usually poor, with about 30 per cent of the conveyor places requiring double timber and 25 per cent requiring taking down 6 in. or more of roof rock. Owing to reduction in width of working places, a substantial amount of timber was saved as

compared with hand work. Systematic propping on at least 6-ft. centers was required in all places. No satisfactory solution was found to the problem of transporting timber from the entry to the working face. Most of the working faces required close timbering and most of the timber was carried by hand from the entry to the working face. In some places, small timber trucks were used, pushed on wheels on the bottom of the trough line. Experiments have been made on timber trucks operated by the trough-line motion. It is the writer's opinion that this problem has not been solved satisfactorily and warrants further study, as too much productive working time is now consumed transporting timber from the entry to the working face. At least one manufacturer has a reversible unit now on the market, which, it is claimed, will transport timber to the working face, but with this the writer has had no experience.

Permissible explosives and electric firing were used throughout the mines. Pillar miners were allowed to fire three holes at one time and solid miners an entire cut. Chemical delay igniters were used for firing a series of holes.

Main haul transportation was almost entirely by rope haulage, each shaft level receiving coal from a hoisting slope sunk and graded in the bed. Gathering transportation was 75 per cent by trolley locomotive and 25 per cent by mules. Mine-car complement consisted of eleven hundred 3.3-ton wooden cars. Mine rock loaded was hoisted and dumped on the surface at the rate of 80 cars per day. About 30 cars of rock per day were unloaded inside by hand. The use of shaker conveyors permitted the gobbing of a great deal of mine rock and refuse in the working places, reducing the amount of rock load at least 50 per cent.

The mine was gassy and all underground men wore electric cap lamps. Each miner was required to keep a Kohler safety lamp lighted at the high point in his working face and to test for gas after each blast.

During the period covered by this paper, an 8-hr. day prevailed for 4½ yr. and a 7-hr. day for 6 months. No change in producer performance on conveyor mining was noted in the change from 8-hr. to 7-hr. day. A decrease of 0.33 tons per man-day in hand mining occurred in the shortening of the working day. Producers spent their full working shift at the face if necessary to produce their quota of tonnage. There were no union restrictions on the amount of tonnage loaded. Individual performance up to 16.5 tons per man-day in the best workings was obtained.

An attempt was made to use duckbills on the conveyors, for loading, but they were found unsatisfactory for low veins and also for pillar mining because of the close timbering required. Very good results were obtained in mining a small solid area where the vein was clean and 5 ft. thick, with a good roof.

The mine in question had been worked formerly entirely as a hand-loading mine, with all mine cars delivered at the working face. This method of work was too expensive and with the drop in realization on anthracite the mine was forced to close in 1932. After being idle for 7 months, the property was reopened and worked for 5 yr. by the methods outlined. The daily production was increased 100 per cent and operating costs were reduced 36 per cent. More difficult mining conditions and further decrease in realization, together with increase in wage rates and payroll taxes, forced the mine to close again in 1937.

Performance Obtained from Shaker Conveyors

The figures in Table 1 on performance of shaker conveyors under various beds and conditions mined include all the work done in the conveyor place performed by the conveyor miners and laborers, starting with the installation of the conveyor and ending with the removal of the conveyor from the working place. The time required to install, remove and repair the conveyor is included, together with time spent timbering, re-timbering, cleaning through caved opening, taking down top rock, moving flush and gob, and any other work required in driving the conveyor place or removing the pillar. For purposes of comparison, the performance from hand-loading direct into mine cars is shown in similar or better conditions. The best and poorest performance for any year and the average performance for the 5-yr. period are given. For detailed description of the conditions existing in each bed and area, see the Appendix.

A substantial increase in producer performance is noted in all veins. The best results are shown in the thinner veins; namely, the Five-foot, Top Ross, Bottom Ross and Eleven-foot No. 2. The cave conditions in the Four-foot and Six-foot veins account for the substantial difference between hand and conveyor work. Similarly, in Eleven-foot No. 3, owing to flushed conditions, conveyors show a substantial increase over hand work. The Eleven-foot No. 1 shows that conveyors do not have a decided advantage over hand loading under conditions such as existed in this area; however, here the result is distorted by the fact that conveyors worked areas where conditions were much poorer than hand-loading areas. The same situation prevails in the Bottom Ross No. 2 where, in addition to poorer mining conditions, a considerable amount of development work in caved ground was done by conveyors. The small difference in performance in the Red Ash vein was caused by conveyors working conditions that could not be worked by hand loading.

Machine Costs

The cost of operating shaker conveyors is a more substantial item than is generally recognized, particularly when the first cost and short

TABLE 1.—*Performance of Shaker-conveyor Places as Compared with Hand-loading Places*

TONS PER MAN-DAY, ALL DEAD WORK INCLUDED

Bed	Bed Section	Tonnage		Number Years Worked	Tons per Man-day						
					Best Year		Poorest Year		Average		Increased by Conveyor
		Hand	Conveyor		Hand	Conveyor	Hand	Conveyor	Hand	Conveyor	
Top Five-foot[a]	Coal, 2'8" Rock, 1'10" Total, 4'6"										
Bottom Five-foot[a]	Coal, 2'8" Rock, 0'4" Total, 3'0"	39,517	292,785	4	5.67	7.24	4.82	6.64	4.88	7.13	2.25
Four-foot	Coal, 4'8" Rock, 0'4" Total, 5'0"	137,233	243,315	5	6.50	9.87	5.18	6.54	6.12	7.54	1.42
Six-foot	Coal, 4'6" Rock, 0'6" Total, 5'0"	84,067	316,239	5	5.63	9.87	4.13	5.42	5.25	7.43	2.18
Eleven-foot, area No. 1:											
Top[a]	Coal, 2'3" Rock, 0'3" Total, 2'6"										
Bottom[a]	Coal, 2'8" Rock, 1'0" Total, 3'8"	126,073	82,526	5	5.50	6.20	4.52	5.68	5.10	5.92	0.82
Eleven-foot, area No. 2[a]		94,257	203,082	5	4.82	5.93	3.56	5.71	4.30	5.89	1.59
Eleven-foot, area No. 3[a]		59,188	146,741	5	4.32	5.48	1.95	4.92	3.90	5.22	1.32
Top Ross	Coal, 2'2" Rock, 0'6" Total, 2'8"	11,401	150,674	3	6.28	6.28	2.02	4.95	4.06	5.72	1.66
Bottom Ross, area No. 1[a]	Coal, 3'8" Rock, 1'0" Total, 4'8"	57,845	372,771	5	6.08	7.30	3.46	6.16	4.35	6.52	2.17
Bottom Ross, area No. 2[a]		98,927	159,772	5	5.40	6.23	3.36	5.22	4.82	5.80	0.98
Bottom Ross, area No. 3		49,991	126,667	3	6.70	7.92	4.53	7.22	5.08	7.62	2.54
Red ash	Coal, 5'0" Rock, 2'0" Total, 7'0"	99,841	20,600	1					4.92	5.58	0.66
Total		1,109,603	2,169,635		5.30	7.30	4.32	5.88	4.85	6.40	1.55

[a] Top and bottom beds mined as contiguous beds over part or all of these areas.

TABLE 2.—*Equipment for Shaker Chute*

Item	Conveyor Troughs	Trough Bolts	Ball Frames	Swivels	Drive Troughs	Drive Jacks	Drive Units	Motors[a] and Switches	Total
New units purchased	2,175	4,350	2,175	87	87	348	87	87	
Replacements purchased	2,548	7,228	1,200	5	16	25		6	
Total received	4,723	11,578	3,395	92	103	473	87	93	
Inventory on hand	1,385	2,320	923	44	84	252	84	86	
Consumption	3,338	9,258	2,472	48	19	221	3	7	
Plus ½ material on hand	692	1,160	461	22	42	126	42	43	
Actual consumption	4,030	10,418	2,933	70	61	347	45	50	
COST									
Per unit	$ 8.40	$ 0.40	$ 5.30	$ 70	$ 40	$ 14	$ 500	$ 150	
Equipment consumed	33,852	4,167	15,545	4,900	2,440	4,858	22,500	7,500	$ 95,761
Maintenance and repairs									
Labor[b]									60,000
Material	700		300	600	500	910	9,000	12,000	24,010
Total	$34,552	$4,167	$15,845	$5,500	$2,940	$5,768	$31,500	$19,500	$179,771

[a] Second-hand motors used.
[b] Included inside inspection, wiring and repairs, also shop repairs.

life of this type of equipment are taken into consideration. Table 2 shows the cost of material and the amount used for 87 shaker conveyors. Depreciation of 50 per cent of equipment on hand at the end of the 5-yr. period is allowed. All of the 87 units were not purchased at one time, the purchasing being spread over a 4-yr. period.

Table 2 includes material lost through caves and other reasons. Two complete units were lost in caves and another damaged beyond repair. A considerable amount of troughing, a number of ball frames and swivels were lost in caving on pillar faces. Every effort was made to keep material cost as low as possible. Damaged trough bolts and troughing were carried outside; the bolts were straightened and rethreaded, the troughing was straightened and patches were welded on. Good sections of troughing were cut off and welded to other sections, making one renewed trough out of two discarded ones. When the troughing was too far gone for repairing, the attachments were burned off and welded onto new steel plates, which were purchased already formed. Parts for swivels and jacks were purchased and damaged or wornout swivels and jacks were rebuilt. The balls in ball frames were renewed until the frames were worn through. Puller rods, drive troughs and double swivels were straightened and re-bushed.

A wide variation in maintenance costs between different drive units was noted. The better type could be maintained after they were 2 or 3 yr. old for an expenditure of $30 to $60 per year for repair parts per drive unit. The poorer models cost several times this amount after they had been in service for periods of from 1 to 2 yr. The maintenance cost of electric motors was too high, largely due to the second-hand equipment purchased.

The total tonnage mined during the period by the 87 units amounted to 2,169,635. The number of machine shifts worked during the period covered was 129,972. The cost of operating and maintaining conveyor equipment per machine shift worked amounted to $1.3831 per shift. The average amount produced per machine shift worked was 16.69 tons.

The number of tons of coal produced per unit of equipment consumed is shown in Table 3.

TABLE 3.—*Consumption of Shaker-chute Equipment*

Item	Number Used	Tons Mined per Unit of Equipment Used
Conveyor troughs	4,030	538
Trough bolts	10,418	208
Ball frames	2,933	740
Swivels	70	30,994
Drive pans	61	35,567
Jacks	347	6,253
Drive units	45	48,214

Conclusions

1. The better grade of shaker-conveyor equipment on the market is well designed and efficient, with the exception of the trough swivels. Manufacturers of shaker conveyors should develop a more practical and efficient type of swivel and should experiment on developing a better ball frame.

2. Shaker-conveyor mining in low and in caved and flushed veins showed an average increased producer performance of 1.55 tons per man per day, as compared with loading direct into mine cars; also a substantial saving in transportation cost. Mining conditions were better in hand work, and under equal conditions it seems probable that the conveyor work would have shown a further increase in performance.

3. The cost of purchasing and maintaining conveyor equipment amounted to $0.0826 per ton produced and $1.383 per machine shift worked.

4. Conveyor miners were able to maintain their performance with reduction in working day from 8 to 7 hr. Performance of hand miners dropped 8 per cent with the shorter working day.

5. The life of the mine was extended for 5 yr. through economies largely effected by shaker-conveyor mining, and nearly 4,000,000 tons of coal was recovered, a large part of which could not have been mined by former methods.

APPENDIX

Detailed Description of Mining Conditions in Various Beds Shown in Table 1

Top and Bottom Five-foot Beds.—These two beds are contiguous and separated by 4 to 8 ft. of rock interval. As they are the surface beds,

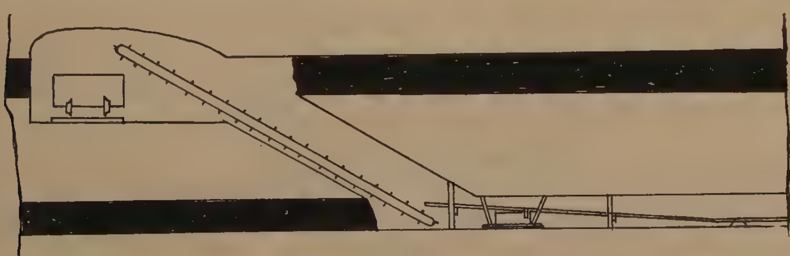

Fig. 3.—Development in top bed, mining contiguous seams.

only first mining was done in them, owing to shallow rock cover. The development work was driven originally in the Top Five-foot bed. In order to eliminate development expense, the lower, or Bottom Five-foot bed, was mined through 30° rock holes driven down from the top bed as shown in Fig. 3.

Chain conveyors were installed in these rock holes to elevate the coal into mine cars in the upper bed. Holes were driven 300 ft. apart and shaker conveyors were installed on the entries in the lower bed to receive coal from the chamber units. From three to four conveyors were used in each hole.

A typical bed section of both veins is shown in Table 4.

TABLE 4.—*Typical Bed Section, Five-foot Beds*

	Top Five-foot Bed			Bottom Five-foot Bed		
	Coal	Refuse	Total	Coal	Refuse	Total
Bone		1′6″	1′6″			
Coal	1′0″		1′0″	2′0″		2′0″
Rock		0′4″	0′4″		0′4″	0′4″
Coal	1′8″		1′8″	0′8″		0′8″
Total	2′8″	1′10″	4′6″	2′8″	0′4″	3′0″

The Top Five-foot bed was mined largely by hand loading. In the hand places the top bench of bone was taken down and gobbed to make height for cars. In conveyor places this bench was propped and held up. The performance of men working on coal only in this bed on hand work varied from 5.1 to 6.6 tons per man-day. The performance on conveyor work, men working on coal only, varied from 6.0 to 8.5 tons per man day. The Bottom Five-foot bed was mined entirely by conveyors. The performance in this bed by conveyor, men working on coal only, varied from 6.6 to 10.2 tons per man-day, the bottom bed being much easier to cut coal in and having better roof conditions, thus giving the better performance. The production records for these beds were not kept separately.

Four-foot Bed.—The Four-foot bed had some of the best mining conditions at the colliery. The roof was good and the vein open in 60 per cent of the work. The remaining 40 per cent was caved or partly caved, and required close propping, and, in a number of places, double timbering and forepoling. A small area from 2 ft. 0 in. to 3 ft. 0 in. thick was mined in the solid. About 10 per cent of the work consisted of skipping pillars without robbing them, owing to lack of adequate rock cover to remove pillars. A typical bed section of the Four-foot bed is shown in Table 5.

The performance from hand loading, working on coal only, varied from 6.6 to 9.9 tons per man-day. The performance from conveyors working on coal only varied from 6.6 to 16.5 tons per man-day.

Six-foot Bed.—Approximately 40 per cent of the Six-foot bed mined was similar to the Four-foot bed. An additional 40 per cent of the bed

TABLE 5.—*Typical Section, Four-foot Bed*

	Coal	Refuse	Total
Coal	0′6″		0′6″
Bone		0′4″	0′4″
Coal	4′2″		4′2″
Total	4′8″	0′4″	5′0″

was heavily caved and in a number of places closed tight, requiring forepoling and double timbering.

There was also a large area where the vein was in two splits with from 1 to 4 ft. of middle rock, which had to be taken with the coal. A large amount of bad roof was encountered and several small areas of solid coal from 18 to 36 in. thick were mined. As in the Four-foot bed, part of the conveyor work consisted of skipping pillars without robbing them. A typical section of this bed would be 4 ft. 6 in. of coal and 6 in. of refuse, but the vein varied so widely from this that no intelligent average can be given. The performance from hand loading, working on coal only, varied from 4.95 to 9.9 tons per man-day. The performance from conveyor loading varied from 4.95 to 16.5 tons per man-day.

Eleven-foot Bed, Area No. 1.—The Eleven-foot vein presented three different mining conditions and therefore has been divided into three different areas. In area No. 1, the top and bottom splits of the vein were separated by a rock parting from 1 to 10 ft. thick, and in this area the two veins were mined together by one of the methods previously mentioned. The area was partly caved and old rooms nearly all partly filled by rock either from falls or gobbed from previous mining. A typical section of the two splits is shown in Table 6.

TABLE 6.—*Typical Bed Section, Eleven-foot Beds*

	Top Eleven-foot Bed			Bottom Eleven-foot Bed		
	Coal	Refuse	Total	Coal	Refuse	Total
Coal	2′0″		2′0″	0′4″		0′4″
Rock					0′4″	0′4″
Coal				0′6″		0′6″
Bone		0′3″	0′3″		0′8″	0′8″
Coal	0′3″		0′3″	1′10″		1′10″
Total	2′3″	0′3″	2′6″	2′8″	1′0″	3′8″

The performance of hand miners working on coal only in this vein varied from 3.3 to 8.5 tons per man per day. The performance of

conveyor miners working on coal only in this vein varied from 4.95 to 9.9 tons per man per day.

Eleven-foot Bed, Area No. 2.—In this area the rock separation increased in thickness up to 30 ft., and the two splits of the vein, where present, were mined separately. Most of the mining was done in the bottom bench because the top bench was not usually minable. Several large areas of solid top coal were mined, vein thickness averaging 24 in. The workings in the bottom bench were partly caved and most of the openings partly filled with gob rock from first mining. The performance of hand miners working on coal only varied from 3.3 to 8.5 tons per man-day. The performance of conveyor miners working on coal only varied from 4.95 tons to 9.90 tons per man-day.

Eleven-foot Bed, Area No. 3.—The conditions existing in Area No. 3 were practically the same as those in Area No. 1 and, in addition, the area had been 70 per cent flushed. In mining in the flushed area it was necessary to drive through the flush in old crosscuts and to drive new crosscuts across the old chambers through the flush for ventilation. The performance of hand miners working on coal only in this area varied from 3.3 to 8.5 tons per man per day. The performance of conveyor miners working on coal only varied from 4.95 to 9.90 tons per man-day.

Top Ross Bed.—Large areas of this vein were mined as a contiguous vein with the Bottom Ross. The records given here were made when the vein was mined separately from the Bottom Ross. The mining was 50 per cent solid mining, 50 per cent pillar removal. The roof was usually fair but in a number of places from 6 to 24 in. of top rock had to be handled. A typical vein section is shown in Table 7.

The bottom bench of bone usually was left down. No hand work was done in this vein other than opening places and removing gangway pillars. All drilling was done with compressed-air drills. The performance of hand miners working on coal only in this vein varied from 3.3 to 6.6 tons per man per day. The performance of conveyor miners working on coal only varied from 3.3 tons to 8.25 tons per man per day.

TABLE 7.—*Typical Vein Sections, Ross Beds*

	Top Ross Bed			Bottom Ross Bed, Area No. 1		
	Coal	Refuse	Total	Coal	Refuse	Total
Coal	0′2″		0′2″	3′0″		3′0″
Bone		0′4″	0′4″		1′0″	1′0″
Coal	2′0″		2′0″	0′8″		0′8″
Bone		0′2″	0′2″			
Total	2′2″	0′6″	2′8″	3′8″	1′0″	4′8″

Bottom Ross, Area No. 1.—Mining conditions varied so widely in the Bottom Ross bed that it is necessary to divide this vein also into three areas. A typical section of this vein is shown in Table 7.

The bench of bone was nearly always taken up over the entire face of both conveyor and hand-loading places, as it was too soft to support timber. In Area No. 1 the workings were about 40 per cent caved and old places were partly filled with gob rock. A large area of pillars in the Top Ross when the vein averages 3 ft. in thickness is included in the production from this area. Also included is a large area where the Top and Bottom Ross veins were mined together with a rock separation of from 3 to 4 feet.

The performance of hand miners working on coal only in this area varied from 3.3 to 6.6 tons per man per day. The performance of conveyor miners working on coal only varied from 4.95 to 9.9 tons per man per day.

Bottom Ross, Area No. 2.—In Area No. 2 conditions were similar to those in Area No. 1, except that generally harder mining conditions prevailed. Roof conditions were much worse with a higher percentage of falls to clean. The vein contained a great deal of bone, sometimes as high as 60 per cent of vein thickness. A large area of Top Ross was mined as a contiguous vein and is included in the production from this area. A great deal of trouble was encountered with squeezes where the Top and Bottom Ross beds were close together.

The performance of hand miners on coal only from this area varied from 3.3 to 6.6 tons per man-day. The performance of conveyor miners on coal only varied from 4.95 to 9.9 tons per man-day.

TABLE 8.—*Typical Section of Red Ash Bed*

	Coal	Refuse	Total
Coal	2′0″		2′0″
Blue coal		2′0″	2′0″
Coal	3′0″		3′0″
Total	5′0″	2′0″	7′0″

Bottom Ross Area No. 3.—Area No. 3 of the Bottom Ross vein was one of the best mining areas in the mine. The roof conditions were usually good and there were few falls in the old places. The vein was cleaner than in either of the other two areas. In several large areas the old rooms had been rock-packed. The performance of hand miners working on coal only in this area varied from 3.30 to 9.90 tons per man-day. The performance of conveyor miners working on coal only varied from 6.6 to 13.2 tons per man-day.

Red Ash.—Very little conveyor work was done in the Red Ash vein; this was reserved for hand work. Several pillars were removed in an area

where conditions were too bad to permit hand mining and a solid area where the vein was from 2 to 3 ft. thick was mined. Work was in progress to install conveyors in some of the better areas. A typical section of the Red Ash vein is shown in Table 8.

About 80 per cent of the old workings had been flushed or filled by rock-packing. The roof conditions were generally fair but it was necessary to stand cross timber in some of the conveyor work. The performance of hand miners working on coal only varied from 4.95 to 9.9 tons per man-day. The performance of conveyor miners on coal only varied from 4.95 to 13.2 tons per man per day.

DISCUSSION

(*W. S. Lesser presiding*)

H. H. Otto,* Scranton, Pa.—The colliery described by Mr. Marshall probably would not have been reopened if it had not been possible to use undercutters, shaking chutes, etc. There are possibilities of increasing the total output from mines in the northern anthracite field by similar methods. The shaker chutes, as modernized, enable mining not only in low seams but robbing in caved and crushed ground, which would be lost if we were obliged to take a mine car into the working face.

J. J. Maloney,† Wilkes-Barre, Pa.—Mr. Marshall brings out the fact that the maintenance cost for the operation of well-designed shaker conveyors is relatively low except for the trough line. The manufacturers have recognized that under operating conditions found in the anthracite mines, trough maintenance is likely to be high. This is brought about by the various degrees of pitches on which the trough lines operate. It is well known that a shaker conveyor with a given motion will operate most efficiently on a grade for which it was designed. For instance, it takes only an agitating or jigger motion to move coal downhill; it takes a moderate, quick motion to move coal on the level, and an intensive, quick, return motion to move coal uphill. Trough lines are operated on from minus 10° against coal travel to plus 30° in favor of coal travel. Mr. Marshall has partly corrected this high trough maintenance by specifying that the trough plates be made from 3/16-in. steel plates instead of 1/8-in. steel plates, which is standard for the smaller sizes of troughing. I believe this is good practice and that the trough will give a much longer life at a slight increase in price. However, this makes the trough considerably heavier to handle and does not correct the condition that causes excessive loads or stresses on the trough line, such as heavy grades or sudden fluctuations in voltage.

In the selection of a shaker conveyor, a great deal of thought should be given to choosing the proper motion to more nearly meet the physical conditions to be encountered. Trough-line stresses increase directly as the square of the speed, and sudden drop or increase in voltage is bound to result in high maintenance cost in the trough line. It is, therefore, quite essential that the voltage at the motor terminals be kept constant. The recent application of flywheel motors to shaker conveyors has cut down trough maintenance by preventing overspeeding and underspeeding of the trough line during reciprocation. This reduces trough-line stresses, resulting in longer service from the trough line.

A uniform grade condition and constant voltage, while seldom found in coal mines, are ideal for shaker-conveyor application.

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* The Hudson Coal Co.

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Recent Trends in Rock Dusting to Prevent Dust Explosions in Coal Mines

By H. P. Greenwald,* Member A.I.M.E.

(Chicago Meeting, October 1938)

Those interested in the early developments and experiments, both in the United States and abroad, that led to modern rock dusting, will find an excellent summary in a paper by George S. Rice,[13] published in 1914. The period from 1914 to 1924 was one of intensive experiment on coal-dust explosions in the United States; abroad, war activities and rehabilitation stopped this work over a large part of that decade. American findings were published by the Bureau of Mines, particularly in *Bulletins* 167 and 268.

It is convenient to consider that in the United States rock dusting to prevent coal-dust explosions in mines was initiated in 1924 with the publication of tentative specifications.[17] This was followed in 1925 by an investigation of methods and costs[12] and approval by the American Engineering Standards Committee of a Recommended American Practice.[18] Approximately 13 years have elapsed since this approval, and inasmuch as changes in rock-dusting practice in the United States have been evolutionary, it is necessary to consider this entire period as "recent" in some phases of the work. There have been no official changes in the Recommended American Practice since its approval.

This paper reviews as briefly as possible knowledge acquired since the issue of the earlier papers mentioned, through experience in mines and experiments conducted by different organizations engaged in the promotion of mine safety.

Size of Rock Dust

The Recommended American Practice states that the dust shall be 100 per cent minus 20-mesh and 50 or more per cent minus 200-mesh. Results of large-scale tests on the relative effectiveness of rock dust of different sizes were published by the Bureau of Mines[15] in 1933 and are reproduced here as Fig. 1. The limiting curve of this figure shows that

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[13] References are at end of paper.

there is advantage in finer grinding up to 70 per cent minus 200-mesh; beyond this there is no additional advantage. Extremely fine dusts cake more easily than coarser dusts, and this is another reason for avoiding them.

Results of laboratory tests published by the Bureau of Mines[4] in 1935 indicated slightly less increase in effectiveness with increase of fineness of rock dust than did the above-mentioned large-scale tests. However, the difference is little more than the error of duplicate experiments and is negligible in so far as application to mining is concerned.

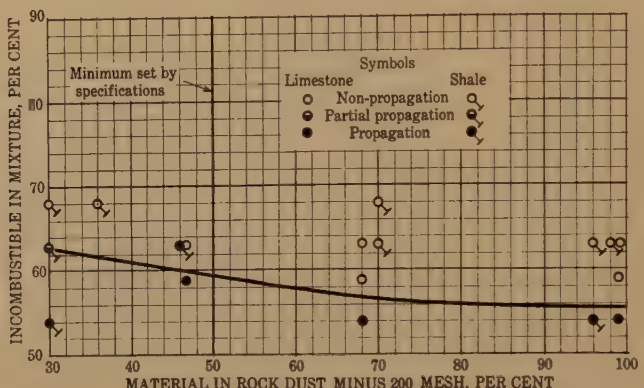

FIG. 1.—RELATIVE EFFECTIVENESS OF ROCK DUSTS OF DIFFERENT SIZES.

The characteristics of modern grinding machinery make the stipulation of 100 per cent minus 20-mesh superfluous in most instances. Dust that is 70 per cent minus 200-mesh will usually run over 99 per cent minus 65-mesh.

Composition of Rock Dust

The Recommended American Practice placed only two restrictions on composition—maxima of 5 per cent combustible material and 25 per cent free silica. Five per cent combustible material is a liberal allowance for rock dusts available in most mining districts, particularly if the dust is limestone. Dusts containing not over 2 per cent are available in most places, and less than 1 per cent is not uncommon. Obviously, if two dusts are otherwise equal, the one containing least combustible matter is preferable.

The limit of 25 per cent on free silica is doubtless too high in view of modern researches on silicosis, and mining men should be prepared for a drastic revision downward when studies now under way begin to bear fruit. In July 1935 the Public Health Service advised that if the mine personnel is repeatedly exposed to breathing the dust, the percentage of *free and combined silica* be as little as possible, preferably *less than* 5 *per*

cent. This restriction is not burdensome in districts where pure limestones or gypsum are available locally. In some districts, strict compliance with this recommendation may mean increased freight charges because of the additional distance that suitable dust has to be transported. To what extent use of rock dust creates a health hazard still remains to be determined accurately. In the meantime, common sense dictates that men operating rock-dusting machinery or applying rock dust by hand should wear respirators.

Caking of Rock Dust

The Recommended American Practice states that rock dust shall not absorb moisture from the air to such an extent as to cake or destroy its effectiveness as a dry dust. It should be noted that no dust likely to be used for rock-dusting purposes absorbs large quantities of moisture from unsaturated air as, for example, does calcium chloride. The moisture content of a dust always tends to be in equilibrium with the humidity of the air with which it is in contact. If the humidity of the air rises, moisture content of the dust rises, and vice versa. However, even with atmospheres close to saturation, the dust will not become truly wet, although sometimes there may be a noticeable tendency toward agglomeration.

In so far as effectiveness against coal-dust explosions is concerned, a greater change results from absorption of moisture by rock dust when brought in contact with a wet surface or when moisture is deposited on it directly from a supersaturated atmosphere. Depending on conditions, rock dust subjected to such treatment may vary from a compact mass to mud. This may happen in a mine entry that is naturally wet or in places where moisture is deposited during warm weather. The ability of coal dust to resist wetting is well known, but if it is mixed with rock dust the whole mass becomes wet. Also, a thin sprinkling of coal dust over wet rock dust will become bound to the rock dust to some extent, but the top of a thick layer of coal dust so deposited is likely to remain dry and readily dispersible, whereas it will not be possible to form a cloud of the rock dust beneath.

The condition of rock dust that has been wetted and dried varies widely, depending on the dust. Some kinds will be nearly as dispersible as when first ground, and others will be caked into a solid mass that cannot be considered as dust. The requirements of the Recommended American Practice made it necessary that the Bureau of Mines develop a test for grading the relative caking of rock dusts when wetted and dried. A stiff paste is made by mixing water with the dust under test, and this is packed into a mold. The molded form is then thoroughly dried and crushed in a small testing machine. Molds of feebly caking dust disintegrate under low pressure. On the other extreme, a few dusts have

caked so badly that their strength when dried exceeded the capacity of the machine. Whether or not a dust is suitable for use in coal mines is judged by comparing its strength in this test with that of other dusts of which the behavior underground is known. Pure high-calcium limestone dusts and pure dolomite dusts show least caking. In general, caking increases with increase of impurities in these dusts. Gypsum and shales cake badly.

In connection with tests of barriers, there was brought to the Bureau's attention recently a limestone dust that is treated with a water-repellent substance.[5] Apparently each individual particle is coated and the dust is as difficult to wet as is coal dust. Also, if wetted by vigorous agitation with water, it dries without caking. This development promises to be important in locations where rock dust is subjected to dampness.

Dispersibility of Rock Dust

The term "dispersibility" refers to the ease with which a mass of dust at rest can be raised and scattered in air. It was mentioned above that rock dust exposed to highly humid (but not supersaturated) atmospheres may absorb enough moisture to develop a tendency toward agglomeration. This will be evidenced by ability to mold the dust by pressure of the hands, but the molded form falls apart when disturbed. Such dust will require a stronger blast of air to disperse it as a cloud than does thoroughly dry dust. However, the author is not aware of any case in which extended propagation of an explosion in a coal mine in the United States was judged to be due to poor dispersion of agglomerated rock dust.

By contrast it should be noted that in England the chief inspector of mines considered that damp rock dust permitted extended propagation in two explosions. The first of these was at Haig pit, Whitehaven colliery, Cumberland, on Jan. 29, 1931. Concerning it the official report[24] says: "The explosion was spread throughout the district because of dampness causing the limestone to bind and not rise and intermingle with the fine coal dust which lay upon it." The second explosion was at Wharncliffe Woodmoor No. 1, 2, and 3 colliery, Yorkshire, on Aug. 6, 1936. In the official report[25] the chief inspector refers to the above-quoted statement and adds, "I believe this Wharncliffe Woodmoor explosion spread along the haulage roads for the same reason."

Following the Haig pit explosion, the Safety in Mines Research Board of Great Britain inaugurated a laboratory study of the dispersibility of different rock dusts and the effect thereon of exposure to humid atmospheres. Progress of the investigation can be followed in appropriate sections of the Board's reports from 1931 to 1936, inclusive.[19] The report for 1937 is not available at the time of writing.

The more important findings of this investigation were described by Tideswell and Wheeler[22] in a paper presented before the Midland Insti-

tute of Mining Engineers on Oct. 7, 1937. Two tests, designated as *C* and *E*, are described. In test C, a sudden blast of air is directed vertically downward from a rose jet onto a layer of dust in a tray; this test has been of value in estimating loss of dispersibility resulting from adsorption of moisture. Test E makes use of a wooden gallery (Fig. 2). Fig. 3 shows some of the results obtained. By permission, these figures are reproduced from the Board's 15th Annual Report (1936). In the

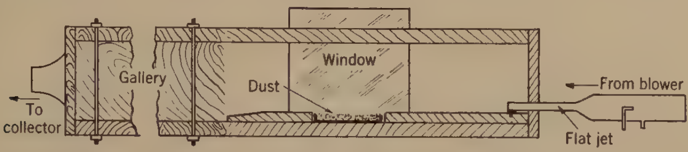

FIG. 2.—GALLERY USED IN BRITISH TEST E ON DISPERSIBILITY OF DUSTS.

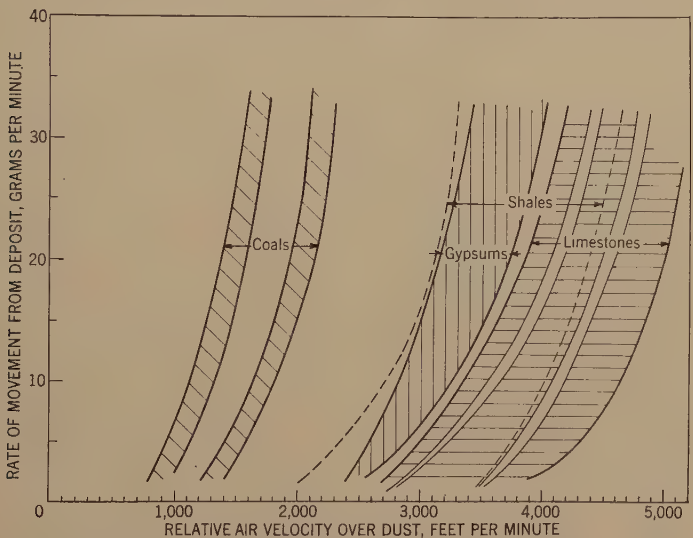

FIG. 3.—RESULTS OF BRITISH TEST E ON DISPERSIBILITY OF DUSTS.

apparatus shown in Fig. 2, a blast of air from a flat jet plays across the surface of a dust deposit in a tray set in the floor of the gallery. The velocity of the air is measured before it enters the jet. In Fig. 3 the rate of erosion of various dry dusts is plotted against air velocity. There is a certain threshold velocity below which practically no dust is moved; above this, the rate of movement increases rapidly. Results fall in broad bands. Coals move far more readily than the rock dusts tested, and English gypsums more readily than limestones.

Tideswell and Wheeler apply the term "weathering" to exposure of a dust to air prior to test. Trays of dust were exposed for varying periods in air of 95 per cent humidity at 77° F. Only gypsums lost their dispersibility markedly, and no difference was observed between dolomitic

and calcareous limestones. In all this work, it was important that the dust be deposited in the tray by a standard method and remain undisturbed until the test was run. The tests prove that dispersibility is a property of the mass of dust and not of the individual particles that compose it. Therefore, any test of the dispersibility of dust in a coal mine must be made on that dust in place in the mine. Blowing sharply and diagonally on the dust is a rough-and-ready test. The authors describe and illustrate an apparatus for reducing the personal error in this procedure, but a truly quantitative measure of the dispersibility of dust in a mine apparently remains to be devised.

Under the cooperative agreement in effect between the Bureau of Mines and the Safety in Mines Research Board, samples of limestone and gypsum used at the Experimental Coal Mine were forwarded to Sheffield, England, for examination by test E. Results obtained with the American limestone fall within the broad band of English limestones shown on Fig. 3, but toward the less dispersible side thereof; also, the loss of dispersibility on weathering was the same as for English limestones. When dry, the American gypsum dust was far less dispersible than the English and was no better than the American limestone. When weathered, very little of the American gypsum was moved, even with velocities exceeding 5000 ft. per minute, which classified it as a decidedly inferior dust in so far as dispersibility is concerned.

It is evident from this that the English results cannot be applied en masse to American mines, and a more extended study of the dispersibility of American rock dusts is desirable. It must be noted also that as yet there has been developed no means of correlating a laboratory dispersibility test with the results of explosion tests at the Experimental Coal Mine, nor with the behavior of rock dusts in explosions in commercial mines.

Dusts Suitable for Rock Dusting

If the more recent suggestions of the Public Health Service are followed, dusts containing silicates other than as minor impurities may not be used for rock-dusting purposes, and only fairly pure sulphate and carbonate dusts are suitable. Of the sulphate dusts, only gypsum and anhydrite have commercial possibilities, and to date their use in the United States has been limited in comparison with limestone because of higher price in most places. Both American and British experiments indicate that limestones are preferable to gypsum and anhydrite for use when places are damp or where high humidity exists, because the limestones cake less and are much more dispersible after exposure to a humid atmosphere.

The process of waterproofing the individual particles of a rock dust (mentioned above) appears to promise the best solution of difficulties

due to loss of dispersibility. If it can be applied to all kinds of carbonate and sulphate dusts, they become equal in so far as this point is concerned.

Incombustible Content Required in Mine Dust

Whether a mine is effectively rock-dusted is judged by the incombustible content of the mine dust, and the requirement varies greatly with a number of factors.[14] Because of this, and because conditions in mines change continually in unpredictable ways, it is impractical to specify in detail the percentage of incombustible required in even a small portion of the cases that are met. Instead, the specification must cover the most severe conditions that are likely to occur, with some factor of safety, and there is then a larger factor of safety for all less severe conditions.

The Recommended American Practice states that rock dusting is to be done so that in the absence of firedamp the incombustible content of samples collected is not less than 55 per cent; this includes any incombustible material inherent in the coal dust itself. An additional 10 per cent incombustible material is specified for each 1 per cent of firedamp in the air current. These minimum amounts are not sufficient to prevent propagation of an explosion under the conditions used in the tests at the Experimental Coal Mine, nor do they make any provision for the varying explosibility of dust in mines working different coal beds ranging in classification from semianthracite to lignite. Table 1 gives a set of minimum figures that take into account the composition of the coal and a more accurate determination of the effect of firedamp in the air current. These are based on tests at the Experimental Coal Mine (ref. 14, pp. 19–24). Intermediate values may be obtained by straight-line interpolation.

Table 1.—*Percentage of Incombustible Material Required in Coal-mine Dust to Prevent Propagation of an Explosion under Conditions of Test in the Experimental Coal Mine*

Volatile Content of Coal, Moisture-and-ash-free, Per Cent	Percentage of Incombustible Required when There Is in the Air Current		
	No Firedamp	1 Per Cent Firedamp	2 Per Cent Firedamp
14	14	31	48
17	31	45	59
20	47	58	68
22	58	66	75
25	61	69	77
40	61	69	77
43	63	70	78
49	69	75	81

Table 1 shows two things: (1) under the test conditions used, the minimum incombustible content set by the Recommended American

Practice is inadequate for coal containing 22 or more per cent volatile matter on a moisture-and-ash-free basis; (2) the amount of added incombustible material required to offset a given percentage of firedamp decreases as the volatile content of the coal increases. Details of the behavior of firedamp in coal-dust explosions are given in Bureau of Mines *Technical Paper* 464.[14]

Experiments conducted by the Safety in Mines Research Board with fine English coal dusts in a cylindrical steel gallery 48 in. in diameter are in qualitative but not quantitative agreement with the data given in Table 1.[10] The relation between required percentage of incombustible and volatile content of the coal reported by the British station is simple and capable of mathematical expression. Laboratory tests (ref. 4, pp. 10–11) have indicated that no similar relation may be expected with American coal dusts of the size commonly found in mines.

According to the Recommended American Practice and the data of Table 1, all kinds of rock dust are equally effective in limiting coal-dust explosions, and no differences of importance have been found between the few rock dusts used at the Experimental Coal Mine. Shale was used in the earlier tests and limestone more recently, and such difference as exists in their effectiveness appears to be due to difference in size, as Fig. 1 shows. Tests made in 1936 showed that gypsum was somewhat superior to limestone under one test method, but not under two others. There was nothing in the data to justify the permitting of a lower incombustible content in mine dust when gypsum is used. An attempt was made to determine the effectiveness of common salt, but the cheap grade used agglomerated rapidly into a solid cake after it had been reduced to requisite fineness.

These results are at variance with the data of similar tests made by the Safety in Mines Research Board[9] under conditions mentioned above for different coal dusts. Gypsum was found to be decidedly more effective than limestone, and its superiority was ascribed to combined water, of which it contains 8.4 per cent. This conclusion was substantiated by tests in which Epsom salts and borax were used as rock dusts; these contain, respectively, 14 and 13 per cent combined water, and were superior to gypsum. At present there can be offered no explanation of the discrepancy between the results of these American and British tests. The American tests show that the gypsum used at the Experimental Coal Mine is not superior to limestone under all conditions, and safety demands that no preference be accorded it.

Determination of Incombustible Matter in Mine Dust

If the rock dust used in a mine is white (as are most limestones and gypsum), a person making observations at regular intervals soon learns to estimate incombustible content with some accuracy through the

change of color resulting from deposition of fresh coal dust. Such observations may serve as a guide in deciding where samples for more accurate determination should be taken. Sampling must be accurate, and a standard method such as that developed by the Bureau of Mines[11] should be used.

Standard methods of chemical analysis may be used for determination of incombustible content, but more time and expense are involved than is justified in many cases. Care must be taken also to use analytical methods adapted to the dust under test. For coal dust alone, the percentage of total incombustible can be taken as the sum of moisture and ash as determined by the methods of proximate analysis used for coal. These methods cannot be used with mixtures containing limestone or gypsum. Limestone decomposes, with evolution of carbon dioxide, when heated at temperatures used for determination of ash in coal. Gypsum contains chemically combined water, which is driven off in part at 105° C., the temperature used for determination of moisture in coal.

Use of a volumetric method is more rapid and sufficiently accurate. About 15 years ago the Bureau of Mines modified the Taffanel (French) volumeter to adapt it to American conditions,[2] and an outfit, including equipment for taking samples underground, has been on the market nearly as long. Operation of the volumeter depends on the difference in gravity between coal and rock dusts. A given weight of coal dust displaces a certain amount of liquid, an equal weight of rock dust displaces a smaller volume, and the displacements of mixtures fall on a straight line connecting the extremes. The volumeter must be calibrated for the coal dust and rock dust of the mine at which it will be used, and the calibration can be either in percentage of rock dust or in percentage of total incombustible in the mixture. The latter is preferable.

About three years ago there was developed in Germany a method of estimating the incombustible content of mine dusts by comparison of their color with standard colors. The apparatus is compact and is carried underground. The comparison is made on a very small sample, about as much as one would pick up on a knife blade. The comparison takes only a minute or two, and the person carrying the instrument underground is supposed to take samples frequently as he travels. This introduces a large personal equation into the sampling, which is undesirable. It is understood that this method is used widely in Germany and may be used to some extent in France, but the author has no record of its being used in England and his opinion, after examining one of the instruments, is that it will not be found satisfactory in American mines.

In Great Britain the Safety in Mines Research Board has developed a method for routine determination of the inflammability of mine dusts[3] instead of analyzing them. The dust to be tested is blown through a

heated tube after standardization of the equipment under carefully controlled conditions. If white flame is projected from the end of the tube the dust is inflammable. If the sample is close to the border line of inflammability, a series of five tests is made to fully determine its characteristics. An apparatus of this type installed in a colliery laboratory may be calibrated by test of dust mixtures of known inflammability furnished by the Board. If a mine dust is inflammable, it may be mixed with different proportions of inert dust, and the amount thereof required to render it inert may be determined. The Board has published no information as to the extent to which this practice has been adopted by the mining industry, nor as to experience with it since 1934. General experience with laboratory testing of the inflammability of coal dusts indicates that the details of the test method must be controlled rigidly if erroneous results are to be avoided.

Methods of Applying Rock Dust

Rock dust may be distributed by air current, by hand, and by machine. Distribution by air current is usually not satisfactory, because a large proportion of the rock dust is deposited close to the point at which it is thrown into the air, and more distant points are inadequately treated.

Rock dusting by hand can and should have a definite place in protecting every mine against explosions. It should be used to keep rock dusting close to advancing faces between the visits of the rock-dusting machine. It can also be used at points inaccessible to the machine. Rock dusting should be kept close to all working faces, for it is at or near such faces that explosions are most likely to start. If a considerable length of passageway intervenes between the point of ignition and the end of rock dusting, an explosion may develop sufficiently to travel a long distance into the rock-dusted portion of the passageway. Not having rock dust close to the working faces may be likened to allowing a fire 15 minutes in which to develop before an attempt is made to extinguish it.

Wherever track and power are available, one of the various types of mechanical distributors is the method commonly used for distributing rock dust. While they differ in detail, the basic principle of all is the same: a high-velocity air current is induced by a fan or blower into which dust is fed mechanically. Discharge is either through fixed openings or through a flexible pipe, which makes it possible to direct the stream to different points. High-pressure machines have blowers capable of forcing the rock dust through hose lines for rock-dusting trackless entries or other places that the machine cannot enter. Rock-dusting trackless entries is the most difficult part of the job, and most likely to be neglected. Such neglect invites disaster. Rock dust in one

entry will not stop flame traveling in a parallel entry that has not been rock-dusted.

Rock-dust distributors should be permissible machines, otherwise they may ignite an accumulation of gas, as is known to have happened in one case.[23] Permissible distributors marketed by three different manufacturers are described in Bureau of Mines *Report of Investigations* 3345.[8]

Mechanization has made necessary the development of special rock-dust distributors for entries in which track is replaced by belts or other forms of conveyors. One manufacturer had such a machine on exhibition at the Annual Convention and Exposition of the American Mining Congress at Cincinnati in May 1938. It is to be expected that there will be considerable development along this line during the next year.

Extent of Rock Dusting in the United States

Statistics on use of rock dust in coal mines in the United States for the years 1930–1933 were published by the Bureau of Mines in 1935.[1] Statistics for the years 1934–1937 are not available at the time of writing but probably will be published in October. Table 2 shows the status of rock dusting on a percentage basis during the years 1930–1933. These figures

TABLE 2.—*Extent of Rock Dusting in the United States*[a]

Year	Percentage of All Mines Using Rock Dust	Percentage of Total Production from Mines Using Rock Dust	Percentage of Total Underground Employees in Mines Using Rock Dust	Percentage of Total Underground Man-hours Worked in Mines Using Rock Dust
1930	8.4	33.2	27.5	30.1
1931	8.3	31.9	26.8	29.1
1932	7.5	30.6	25.7	27.1
1933	7.2	30.7	25.4	27.4

[a] From Bureau of Mines *Report of Investigations* 3295.

show that rock dusting was then confined to the larger operations and that over 70 per cent of the man-hours in the industry were worked in mines in which no rock dusting was done. Furthermore, use of rock dust declined somewhat during the period in question, which was one of economic depression and much idleness.

In 1933 the five leading states and the percentage of man-hours worked in rock-dusted mines were: Utah, 84.7; Wyoming, 78.7; Alabama, 68.9; New Mexico, 59.0; Washington, 43.8. Thoroughness of rock dusting may be estimated on the basis of pounds of rock dust used per ton of coal produced in rock-dusted mines. On this basis the order was: New Mexico, 2.58; Colorado, 1.73; Utah, 1.53; Alabama, 1.04; Pennsylvania, 1.00. The average for the United States was 0.76. The Department of

Mines of West Virginia now issues a monthly mimeographed report on use of rock dust in the coal mines of that state. Rock dust was used to some extent in 156 mines in the state during 1937, with an average of 1.30 lb. per ton. In 57 mines the rate was over 1.00 lb., and in 23 mines it was over 2.00 lb., with a maximum of 5.83. These figures show that while excellent rock dusting is being done in a number of mines, the average for the United States is decidedly poor. Concerning this, D. Harrington and others[7] wrote in 1938:

> Numerous mines in the United States have been rock-dusted with the idea of preventing propagation of an explosion; however, only a small percentage of the supposedly rock-dusted mines contain sufficient rock dust to insure any considerable degree of protection should an explosion be initiated. Many coal-mine operators seem to believe that one application of rock dust will render a mine immune from spreading of explosions during the remainder of its life. Other operators apply rock dust to the main haulage entries and hope that if an explosion occurs it will not travel the air courses. Such methods are not only false economy, but the operator who indulges in such practices is likely to receive a rude awakening. Many mines have been worked out without suffering an explosion, but this has been due more to the grace of God than to any precautionary measures taken by mining people.

Explosions Stopped by Rock Dusting

That adequate rock dusting will check coal-dust explosions promptly is amply proved by experience in the industry. In 1932 the Bureau of Mines published[6] an account of 17 cases in which there was no doubt that rock dust had stopped the development of explosions. Investigation of coal-dust explosions in mines indicates that rock dust has saved 100 to 300 lives each year during the past decade.

What rock dust can do to save lives in any one year depends on how many explosions are started in coal mines. During the fiscal years ended June 30, 1934 to 1937, inclusive, the number of explosions in bituminous mines ranged from 13 to 16 a year. An all-time low record of 19 fatalities was established in 1934. The figures for 1935 and 1936 were 36 and 30, respectively, but in 1937 the number jumped to 57, and unquestionably the record for 1938 will be worse. This is a matter for which mining men must accept responsibility, for the record is entirely a result of the precautions (or lack thereof) that are taken against initiation of explosions and their subsequent development. Thus the worst disaster in the fiscal year ending June 30, 1938, was in a mine in which either no rock dusting had been done or else so little that the explosion was able to spread throughout the entire mine. Also, investigations of explosions have shown repeatedly that failure to rock-dust trackless entries adequately has permitted explosions to spread far beyond the point at which they would have stopped had those entries received treatment equal to that on the haulage roads.

Maintenance of Rock Dusting

As noted above, a single application of rock dust is effective only until contamination by coal dust reduces the incombustible content of the mass below the minimum requirement. Another application of rock dust is then necessary. Obviously, it is advantageous to reduce scattering of coal dust to the minimum, and this should start where most coal dust is produced, at the working faces. Keeping the broken coal and cuttings wet during undercutting, blasting, loading and transportation is the best method of accomplishing it. With mechanized loading, dispersion of dust into the air is particularly bad and is concentrated in relatively small areas, thus contaminating the rock dust in that area rapidly. In the last few years, questions have also been raised concerning the health hazards that accompany the breathing of air containing large amounts of coal dust. The obvious solution of the problem is to treat the dust so that it will not rise in the air easily, and water does this more effectively than any other agent available in mines. It is probable that within a few months the Bureau of Mines will issue a mimeographed publication giving in detail present practice along this line.

It must be emphasized that watering is an adjunct to and not a substitute for rock dusting in so far as prevention of explosions is concerned. Water binds coal dust to the larger pieces with which it is mixed in mining and makes it much more difficult for a dust cloud to form in air. However, such a cloud of wet dust can be formed by a heavy concussion (such as accompanies a gas explosion or an extensive fall of roof), and both experience and experiment prove that such a cloud will propagate flame violently.

Prevention of Dusty Roadways

Dust, either coal or rock dust, as a nuisance on the floor of haulage and traveling ways has not been a serious problem in coal mines of the United States, so far as the author knows. Some mines have used calcium chloride on the floor of main haulages to prevent dust clouds following the passing of trips, which might limit the vision of haulage employees, but this practice has, as far as is known, been confined to main intake airways in winter months.

The different conditions of mining operations in British mines have made the presence of loose dust on the floor of some roadways a nuisance, and the Safety in Mines Research Board has conducted an investigation,[20,21] both in the laboratory and underground, on means of fixing such dusts firmly to the floor. It is possible that the results of this investigation may be of service to American mining men in the future. The first step was to investigate means of wetting coal dust. It was found that the addition of 1 per cent of "Perminal W" (a compound

furnished by Imperial Chemical Industries, Ltd.) reduced the surface tension of water so that it wetted coal dust readily. Furthermore, when the water dried, the dust was caked to some extent and would rewet easily on addition of water alone. Chemicals having a similar effect are available on the American market if needed. Addition of calcium chloride to retain the water was tried also with success. Details of the work are given in the papers of references 20 and 21.

Use in Barriers

The use of rock dust in quantities in barriers placed at suitable locations along mine passageways is a valuable adjunct to rock dusting, but it must be kept clearly in mind that barriers are considered secondary to rock dusting. An explosion must travel to a barrier before it can be extinguished, and the travel may be sufficient to endanger personnel at distant points through production of large quantities of poisonous gases. Inspection of barriers in commercial mines has indicated that all too frequently they are not maintained after installation. The mechanical parts must be in good working order and the dust in them dry and free flowing; otherwise they are liable to fail when an explosion reaches them. Barrier installations should receive inspection and maintenance as regularly as equipment used in producing coal.

In 1932 the Bureau of Mines reported on an extended large-scale investigation of the effectiveness of barriers.[16] The barriers tested were described in detail, together with the method of installing them. Certain forms were recommended for general use, others for special use, and some were not recommended. Following this investigation there were no new developments until 1937, when the safety director of a large coal corporation devised a barrier using rock dust in the paper sacks in which it is shipped, with the modification that the weight per sack is reduced from 80 to 50 lb. This barrier was subjected to a number of tests in the Experimental Coal Mine[5] and was successful against moderate and strong explosions. It was evident that the stronger the explosion, the better the chance of success of the barrier, provided it contained a sufficient quantity of rock dust. Dispersion of dust from the opened paper sacks after operation of the barrier units depends entirely on the force developed by the explosion, and with weak explosions this dispersion is poorer than with barriers recommended by the Bureau of Mines.[16] The behavior of all types of barriers against extremely weak explosions has not been thoroughly investigated by test, because such explosions are difficult to control in the Experimental Coal Mine; they tend either to develop strongly or to become extinguished before the barrier is reached.

Summary and Conclusions

Experiment has shown that 60 to 70 per cent minus 200-mesh is as fine as rock dust need be pulverized. If the recommendations of the

Public Health Service are followed, only carbonate and sulphate rocks will be ground for rock-dusting purposes. The only sulphates commercially available are gypsum and anhydrite, which cake badly when wetted and dried. Loss of dispersibility caused by absorption of moisture from air of high humidity has not appeared as a problem in the United States but has done so in England, where extensive investigations have been made. In this country, application of a recently developed method of waterproofing rock dust may be the means of avoiding caking and loss of dispersibility completely. It is now possible to state more accurately the amount of rock dust required in mines working different beds of coal, and the data appear in Table 1. While new methods of determining the incombustible content of mine dust have been developed abroad, none of them appears to be better for American mines than the volumeter method now in common use.

Mechanization has brought with it the problem of rock-dusting entries that have only belts or other types of conveyors in them. Manufacturers are now developing machinery for distributing rock dust in such passageways. All rock-dusting machinery should be permissible.

Rock dusting is not practiced as generally in the bituminous coal mines of the United States as it should be. In 1933, Utah and Wyoming led in percentage of man-hours worked in mines using rock dust, but New Mexico and Colorado led in the thoroughness with which rock dusting was done in such mines. The recent advances in West Virginia are noteworthy, and the reports now issued by the Mines Department of that state are the most complete records available anywhere in the United States.

That rock dust will check coal-dust explosions is beyond dispute. It has saved 100 to 300 lives annually over the past decade. Therefore the operator that does not rock-dust or that fails to maintain rock dusting is simply gambling that an explosion will not be started in his mine. Judicious use of water to prevent scattering of coal dust is a great aid in maintenance of rock dusting.

The safety record of the coal mines of the United States in regard to fatalities from dust explosions lies directly in the hands of those who operate them. The good record made in the year 1934 was partly due to precautions taken and partly to chance; chance has swung away from safety in the last two years and can readily swing much farther if known means of preventing the spread of coal-dust explosions are not employed more widely and effectively.

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DISCUSSION

(*Gordon MacVean presiding*)

C. W. Gibbs,* Pittsburgh, Pa.—At our Harwick mine, on Jan. 12, 1938, we experienced an explosion that cost the lives of 10 of our employees. The explosion was, primarily, gas. The cause of the explosion, accumulation and ignition, could not be definitely determined, although several theories were presented following the investigation by a Commission of State Mine Inspectors accompanied by representatives of the United States Bureau of Mines. It was demonstrated clearly that 100 per cent rock dusting was an essential means of preventing the ignition of the coal dust and the catastrophe that would have resulted had the mine not been rock-dusted. In a distance of approximately 1000 ft. from the point of ignition, all evidence of flame was eliminated and the force was checked within 1500 ft. Seven of the employees were killed by the force of the explosion or flame, three were overcome by carbon monoxide at points not farther away than 4600 ft. At the time of the explosion there were 48 men in the mine and there is no doubt all would have been killed and the mine seriously wrecked had not the mine been completely rock-dusted.

The Harwick mine was the fifth mine in the state of Pennsylvania to receive credit for 100 per cent rock dusting, and it has been the policy of the company to maintain this condition from that time.

J. E. Jones,† West Frankfort, Ill.—In Mr. Greenwald's paper it is enlightening and encouraging to note the trends toward more knowledge and better standards of laboratory and scientific nature. It is not encouraging to note the trend downward in the use of rock dust in the coal mines of the nation. From Table 2 the trend for the four years 1930 to 1933 has been definitely downward. An indefinite trend, yet a condition known to be unsatisfactory, is that so well stated in his quotation from D. Harrington and others (*Inf. Cir.* 7004) regarding insufficient application of rock dust per mine.

Other than methods and costs of application, the thought involved in these three trends fairly well covers the subject of rock dusting. Each must be successfully linked with the others to obtain maximum safety against coal-dust explosions.

In my 23 years of direct responsibility for safety in coal mines, the first two years in regard to prevention of coal-dust explosion were largely upon the scientific phase of rock dust as a remedy. I recall vividly the condemnations and criticisms of such a revolutionary scheme. Suddenly, very suddenly, the application per mine phase became my job. This was with the Old Ben Coal Corporation in 1917. Explosions in the coal mines of the nation were numerous, and Mr. D. W. Buchanan, President of Old Ben, believed in the possibility of success with rock dust. At least two of

* General Manager, Harwick Coal and Coke Co.
† Safety Engineer, Old Ben Coal Corporation.

us were convinced of the value of rock dust to stop the explosion propagation of coal dust, thus preventing a widespread explosion originating from any cause. The known methods of application were hand dusting, shelves along the passage walls and an overhead barrier, impractical for use in our mines. We adopted the first two methods of application and all was well except there was no rock dust. From dust clouds on the highways it seemed there were thousands of tons of fine dust available there, so we had men gather the fine dust into cloth bags. Incidentally, there is not as much road dust available from that source as one was led to think from the depth on the buggy wheels. We soon got our first lesson on agglomeration, as explained by Mr. Greenwald. The dry summer dust was piled neatly on the shelves and thrown by hand on the ribs and roof. In a few days, in the 63°F. temperature of the mine, it was just mud. Of course, it was in the summer time that we did this, the usual time for dusty roads. So much for trend on agglomeration. Our next effort was purchase of the finest grindings from a limestone quarry—the finest of their fertilizer product. We installed some of that on the shelves and used the rest for aggregate in concrete work. That was a very definite trend on sizing. These two can hardly be considered as methods; they were simply trials. Our next effort was to send a few pounds of the 200 or 300 ft. of light, soft, gray shale that overlies our coal seam in Franklin County, to the Bureau of Mines for analysis as to quality. It was found to be very good. This somewhat completed our original lessons in trends on agglomeration, sizing and composition of rock dust, all completed in 1918.

We then purchased and installed at one of our coal mines in 1918 the equivalent of a flour mill, at a cost of $40,000, to grind and pulverize our own shale into 92 per cent minus 255 mesh. This, we knew, was finer than necessary. Such fineness was evidently of value in that moisture penetrated only ⅛ in. forming a crust and keeping dry the material underneath. This shale rock dust still is intact in many places in our mines, and the dust is in as good condition as when first installed in troughs, some as far back as 20 years ago. The Bureau reported that it was exceptionally good rock dust. With this we adequately rock-dusted our 12 mines in the Franklin County district and in 1926 started the rock-dusting of our Glen Rogers mine in southern West Virginia.

Our shelf system was installed in only one location in one section in one mine, our first repetitious system being that of hand dusting. We soon developed the Old Ben overhead concentrated barrier, no part of which dropped over 8 in., except the trigger upon tripping, for haulage roads and the trough barrier for trackless passages. We gradually developed machine rock dusting, thus displacing the high labor cost of hand dusting. Our first effort in this was an electric household vacuum cleaner specially equipped with a 250-volt d.c. motor. It did good work except that an attendant was necessary at the suction end to manipulate the hose for approximate correct ratio of dust and air. Its chief value was in showing that the principle of electric machine rock-dusting was correct from which we designed and built machines that are still in use. Similar machines with improvements were soon available from manufacturers.

By this time, about 1922, and later in December, 1923, when an address was invited by John B. Andrews before the American Association for Labor Legislation, and still later in the summer of 1924, when a series of lectures in the Rocky Mountain district was invited and financed by the Bureau of Mines, Old Ben's progress and success in use of rock dust became nationally and internationally known, resulting in coal-mining commissions from nearly every coal-producing state and nation visiting us and inspecting our rock-dusting methods. Not only were we the pioneers in practical use of rock dust on a large scale, but we became the sales agency for this safety measure.

Rock dust was loaded in bulk in mine cars for the mine where our mill was located, and in railroad cars for our other and neighboring mines. This bulk method resulted in considerable waste, especially in transportation changes and piling for use in the mines. To offset this we adopted the use of cloth bags, these being returned to the mill for refilling. The only objection to our shale dust was its high cost, $8.00 per ton at the mill.

Our trend by this time had reached the practice, due to our efficient machine dusting, of rock-dusting the roof and ribs of all entry passages and omitting the overhead concentrated barrier. Use of the trough barriers on trackless entries gradually diminished, because of fair condition of the machine dusting remaining for considerable time.

All of our mines except one in West Virginia and one in Illinois were on the two-entry system. The rock-dusting of the air course on a two-entry system is relatively easy, since this trackless passage is adjacent to the track entry. However, all of our mines were changed to a three-entry, and later to a four-entry system, making the problem of rock-dusting these trackless entries more difficult.

The rashing or spalling of the coal ribs and roof would gradually diminish the rock-dusted value of these trackless entries, not having been re-rock-dusted since the track, after entry development, had been removed. The need of some simple means of rock-dust protection in these trackless entries resulted in our thoughts, about 1935, being given to the hanging of the filled paper bags, as received from the rock-dust manufacturer, in these air courses equipped so as to gradually liberate rock dust into the air current. Upon becoming emptied in a few days or weeks the bags could be replaced by full ones. If successful, bags at 25 to 50 ft. apart would gradually and automatically rock-dust such entries, but possibly not so well close to the roof. However, our trials were not successful. The arching of the dust prevented gradual flow from the bags. Our next thought related to some means for the air wave preceding an explosion to open the bag. In this we eventually were mechanically successful and our next step was to arrange with the Bureau of Mines to test the plan in its Experimental mine. These tests were begun late in 1936 and later more conclusive tests were made. The latter tests are explained in detail in U. S. Bureau of Mines *R.I.* 3411, "Tests of a Barrier Using Rock Dust Bags," by H. P. Greenwald and H. C. Howarth. Reference is given to this report in Mr. Greenwald's present paper.

Results of the original tests at the Experimental mine were such that installation of the bags was begun in our Old Ben mines in Illinois and our Raleigh-Wyoming-Glen Rogers mine in West Virginia in 1937. Our first method is to protect entry intersections as with a barrier. We plan to connect these installations with the bags, gradually developing from the barrier plan to a continuous plan of installation, average distance between bags to be about 5 ft., or 10 ft. between pairs of bags, as found successful with the maximum of coal-dust hazard that is used in the Experimental mine. Possibly for mines of lesser coal-dust hazard than in Franklin County tests or mine-dust analysis will show greater distance between bags as being protective in such mines, thus inducing trend toward a larger percentage of the nation's rock-dusted mines.

Mr. Greenwald says that the new system was successful against moderate and strong explosions and that further work is needed with respect to weak explosions. The tendency, he concludes, from former trials, is for weak explosions either to develop strongly or to become extinguished before the barrier is reached. We, therefore, consider the new system as revolutionary because of its success in stopping moderate and strong explosions and its simplicity and economy of installation. We hope the further work he suggests can be done in the very near future.

In our Illinois mines, some 5000 bags per mine have been installed and in our Glen Rogers mine in West Virginia some 3500 bags have been installed, these in general being on trackless passages. The installations are somewhat semipermanent, since the rock dust is kept clean and not to be used until need to stop an explosion.

An added safety factor in this new system is the widespread distribution of 50-lb. bags of rock dust throughout the mine to be readily available in case of fire. The value of rock dust to extinguish mine fires is well known and needs no comment here.

Mr. Greenwald mentions development of water-repellent rock dust, which promises to be important where rock dust is subjected to dampness. Possibility of false sense of security is well indicated in his quotation from the English chief inspector of mines. Our chief concern in caking of rock dust is in the warmest time of our summer season. We are concerned with this occasional caked condition on the mine walls and have begun installation of the bags in some of the haulage locations, since untreated dust in the bags remains in a dispersible condition for the short period of dampness even where it cakes on the walls and roof. Water-repellent rock dust or moisture-proof paper bags unquestionably increase the safety factor in damp places.

In conclusion, it is decidedly true that there has been a lack of trends in the last decade or so toward adequate rock dusting per mine and toward full protection of coal-dust propagation hazard mines in our nation. Evidently the most hazardous mines are well rock-dusted, those considered not so hazardous are partly rock-dusted, and those closer to, yet not on the border line of nonexplosiveness, are not rock-dusted at all. Fortunately the 8.4 per cent of all mines using rock dust in 1930 down to 7.2 per cent in 1933, as shown in Table 2, do constitute the mines of greatest explosion hazard. There is need of correction. I am of the opinion this new paper-bag system is a trend in the right direction, encouraging greater protection per mine and greater number of mines protected. Possibly this awakened trend will result in more thought upon the solution of this important problem and a method of even greater simplicity of application.

G. S. Rice,* Washington, D. C.—Mr. Greenwald's paper naturally interests me, because for 50 years one of my principal professional studies, begun long before the Federal Government made its first appropriation in 1908, has been to investigate the causes and means of preventing disastrous coal-mine explosions.

In 1893 I gave a paper before the Illinois Mining Institute on a shot-firing explosion in the shallow nongassy Pekay mine, Iowa, the previous year, and referred to the absence of noncombustible matter, such as shale dribblings from the roof, in this, a newly developed mine, as a factor in coal-dust propagation.

Subsequently I gave evidence before state legislative committees in Indiana and Illinois on several explosions.

Presumably because of the interest I had shown in this subject and in better mining methods to prevent coal losses, Dr. Joseph A. Holmes, first Director of the Bureau of Mines, invited me to enter the new Federal service in 1908 and detailed me immediately to report on the explosion testing just begun in France and Great Britain, following a series of explosion disasters paralleling those in the United States in the preceding 5 years, culminating in the Courrières colliery disaster, France, which killed 1100 men.

Mr. Greenwald's comment that "in the United States rock dusting to prevent coal-dust explosions in mines was initiated in 1924 with the publication of tentative specifications" might be subject to the misinterpretation that rock dusting had begun to be used only after the issuance of the tentative specifications. I served, represent-

* Former Chief Mining Engineer, U. S. Bureau of Mines, retired Oct. 1, 1937.

ing the Bureau of Mines, on the Committee that prepared these and the need was brought about by the various manners of rock-dusting (which had been started in a few mines, notably in a Colorado mine in 1911) and especially to combat the idea that rock-dust barriers should be regarded only as supplemental defenses, as set forth in the specifications.

We had tried barriers, first the Tafanel barrier, in the Experimental mine in the beginning of the testing in 1911–12, when the mine was small, as a convenience. At that time watering and humidifying were the methods employed in many mines, but after testing all known methods of dust-explosion prevention, in 1913 we officially recommended generalized rock dusting as the only efficient method of preventing coal-dust explosions, and in 1914 explosion demonstrations were made for the Institute during its meeting in Pittsburgh.

Some mines in Great Britain had voluntarily adopted "rock dusting" or "stone dusting" as they called it, following the testing in the Altofts gallery in 1908–1909, but it was not until after the war, in 1919, that Great Britain and France compelled by law, "stone dusting," or "schistification," and in Germany not until 1925.

The problem as to what constitutes "dust" confronted me in beginning testing in the former Pittsburgh gallery in 1908–1909, as lack of attention to size of particles in prior testing abroad undoubtedly led to great uncertainty. I tentatively defined dust as that passing through a 20-mesh sieve, which was concluded in later testing to be a practical limit. Of course the relative explosibility of coal dust varies, as Mr. Greenwald says, with the percentage of finer sizes present in the mixture. Similarly, the effectiveness of rock dust increases with finer particles to the point where these agglomerate.

To those interested in the historical advances of rock dusting at home and in Great Britain, I suggest reference to a paper I gave, by request, before the Institution of Mining Engineers of Great Britain in 1923, "American Coal Dust Investigations," with an extended discussion by British investigators and mining men, giving comparison to British tests.

This led to an agreement for a cooperation on research between the Safety in Mines Research Board and the U. S. Bureau of Mines, which was officially confirmed the following year. It involved exchange of personnel, of information and test material, and has proved most valuable for both sides.

The difference in "rock dusts" or "stone dusts" used in Great Britain and the United States arises from the fact that in Great Britain, partly owing to shale being a waste product, it was easy to have it ground at the mine and in some instances underground. The shale in the United States mine roofs is apt to be sandy, or else highly carbonaceous, so not generally suitable, but as limestone is found adjacent to most coal fields and, moreover, the fine material from limestone mills furnishing material to iron furnaces, etc., is available at a relatively low price and is on the market for agricultural purposes, it is naturally the material to turn to. Early observations I made on gypsum being tried in western coal mines were most unfavorable, because of its caking.

As regards dispersibility; I have observed the tests made in Great Britain but am not convinced that a steady blast of air, even if sudden, is equivalent to the pulsation produced by an advance air wave.

Mr. Greenwald has discussed application of rock dust in the mine. Wherever it is possible, I favor use of a rock-dusting machine that blows off the coal dust collected since the previous application, not removed by prior cleaning, before depositing fresh rock dust, starting this blowing at the top of the passageway and proceeding downward. In a Utah mine it was observed that the dust mixture on the roof, projections and timbers was richer in coal dust and successively less so on the ribs and floor. Tests were made in the Experimental mine which confirmed previous views

that the most dangerous position of coal dust was in the high places, hence the merit of mechanical blowing on of rock dust by a hand-controlled nozzle.

Mechanization loading has greatly increased the difficulties from distribution of coal dust in the working places, producing sometimes a cloud of fine coal dust in the air. The best remedy is still a question. Between times of loading it is possible that some suction dévice with collector may be possible.

Conveyors need not stir up much dust except when being loaded or discharging, but they have presented a difficult problem in European mines, where they discharge into "mother" conveyors, and into pit cars.

In France I observed, about 1936, in a mine in the Loire basin, what appeared to be an excellent device for taking off the coal dust rising from discharge chutes of conveyors while loading into cars. The end of the discharge chute was partly confined by curtains drawn across the haulageway and the dusty air was sucked by a small compressed-air propellor-driven fan in the mouth of a long box. The box extended over the pit cars at the inby side of the loading chute. It was about 20 ft. long, the sides made of a coarsely woven burlap. There was a fine spray within the box around the periphery of the blower discharge. The dust in the air escaping through the wet burlap sides of the box was completely brought down; none could be observed outside the box or in the open roadway. The wood bottom of the box sloped down toward the chute end and discharged the condensed spray water and dust in a small stream into the top of the pit cars successively loaded and moved.

This had the double merit of getting rid of the dust advantageously and wetting the dry coal to a limited extent in the cars. It was stated by the management that no trouble was experienced from the coal being too wet when delivered to the screening plant on the surface. A somewhat similar arrangement was later being developed by a British machinery manufacturer.

As one retired from the Bureau, I want to express my appreciation of the fine work done by my former associates in the coal-dust explosion prevention research, both in the studies in the coal mines of the country and in the Experimental mine testing. For the latter I express especially appreciation of the work of Mr. Greenwald, who now directs it, and of H. C. Howarth, who has been in immediate charge of the Experimental mine operation since it was started in 1911. Also appreciation of the work of former Experimental mine associates: Dr. J. K. Clement, physicist; W. L. Egy, physicist; and L. M. Jones, mining engineer, who lost his life in 1916 in heroic rescue work in a mine disaster; as well as many others of the Bureau's staff, who assisted in the furtherance of the work that has given the Experimental mine a world-wide reputation.

Factory Testing of Propeller Mine Fans

By Raymond Mancha*

(Chicago Meeting, October 1938)

The number of installations of propeller mine fans completed during the years of 1936 and 1937 is evidence of the increasing popularity of the propeller fan with the American mining industry. During the period from 1931 to 1936, there were several different low-pressure, low-efficiency propeller fans available, which filled a definite need in replacing inefficient centrifugal fans operated at comparatively low pressure by means of purchased power. Since 1936, however, the trend toward the propeller fan has greatly increased, as propeller fans are now available practically without pressure limitations and capable of efficiencies high enough to warrant replacement of efficient centrifugal fans in many instances.

Few manufacturers of propeller mine fans base their fan-operating characteristics upon similar test methods, primarily because of inadequate facilities for testing in the factory and the need of a test code incorporating a calibrated duct or nozzle that makes possible the testing of large propeller fans quickly, accurately and without prohibitive expense and space requirements. As a result, fan manufacturers have resorted to various methods for testing large propeller fans in the factory, and different yardsticks have produced uncomparable and often inaccurate performance data. It is the purpose of this paper to describe and discuss in detail the test procedure of the Jeffrey Manufacturing Co., which has been proved both accurate and practical.

Fig. 1 shows a plan, a side elevation and two end elevations of the test duct with a propeller fan of the exhaust type in place. Testing propeller fans exhausting permits greater accuracy than testing these fans blowing, because in the former uniform flow conditions are more easily established at Pitot tube stations where fan pressure and air-volume measurements are desired. Any residual rotation remaining in the air discharged from a propeller fan blowing into a test duct must be removed before an attempt is made to obtain accurate pressure and volume readings. Although various devices for straightening the air may be placed between the blowing fan and the volume-measuring station without resulting complications, uniform flow so obtained at the pressure-measuring station necessitates corrective calculations to compensate for duct resistance

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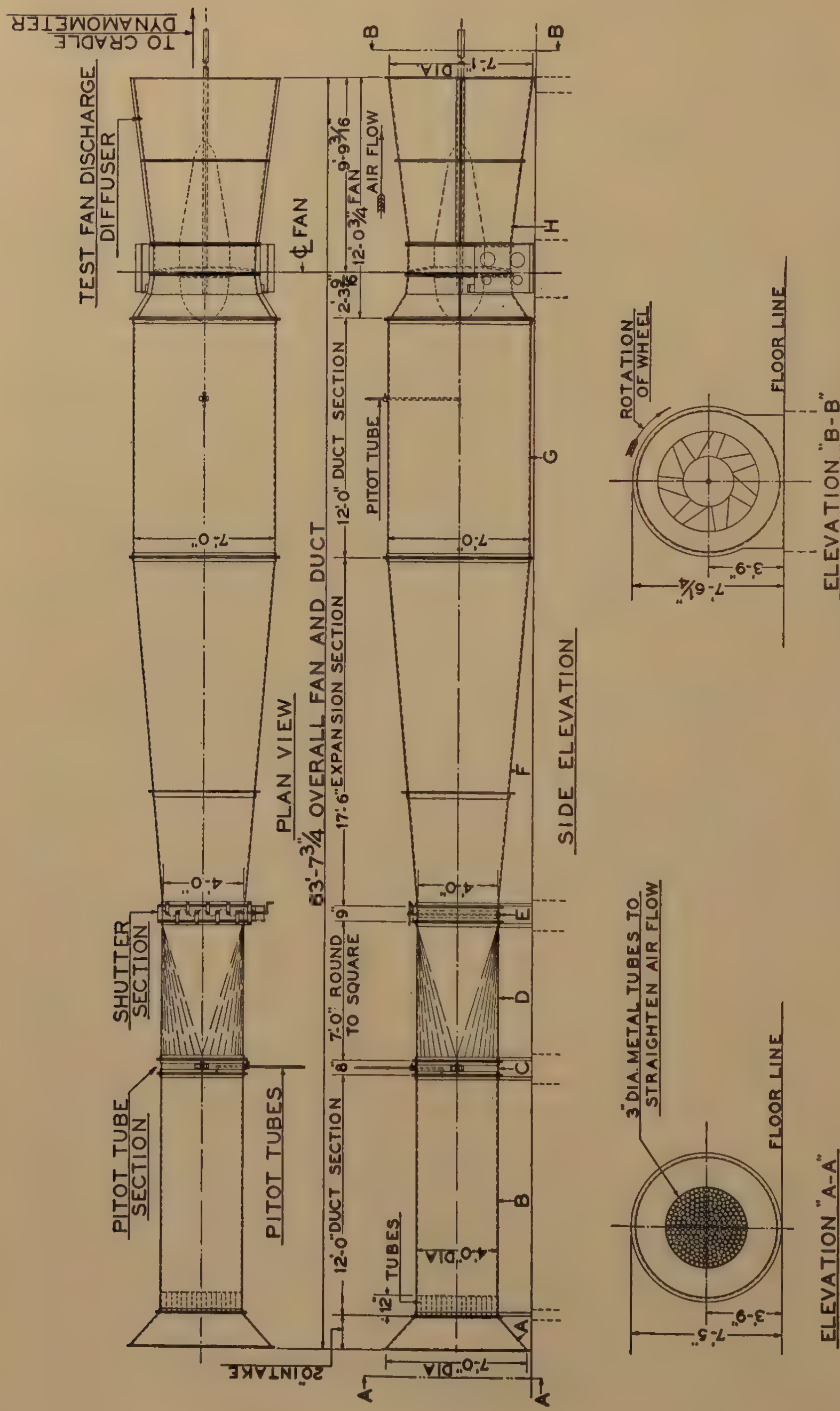

FIG. 1.—AERODYNE FAN AND TEST DUCT.

and resistance offered by the air-straightening device. On the other hand, with the fan exhausting the pressure-measuring station may be placed immediately ahead of the fan and at this point an accurate pressure reading may be read directly without correction for intervening duct friction.

In the duct, section *A* (Fig. 1) is a collector or entrance piece in the shape of a cone frustum 20 in. long, 7 ft. diameter at the point of entrance and 4 ft. diameter at the point of exit. Section *B* is of circular cross section, 4 ft. in diameter by 12 ft. long. At the entrance of section *B* is an "egg-crate" section made up of metal tubes 3 in. in diameter and 12 in. long, tangent to one another. Section *C* is a ring 8 in. long and 4 ft. in diameter in which are situated two Prandtl type adjustable Pitot tubes, one mounted vertically and the other horizontally. Section *D* is a transition piece from round to square, 7 ft. long with a circular opening 4 ft. in diameter at the intake end and a square opening 4 by 4 ft. at the discharge end. Section *E* is square, 4 by 4 ft., and 9 ft. long. In this section eight shutters mounted vertically are actuated by a screw mechanism and hand crank. Turning the crank rotates alternate shutters clockwise and counterclockwise, which controls the volume of air. Section *F* is an expansion section 17 ft. 6 in. long, with an entrance area 4 by 4 ft. and an exit area 7 by 7 ft. This section slopes, so that the corners make an angle of 7° with the longitudinal axis of the duct. Section *G* is a square duct section 12 ft. long and of uniform section area 7 by 7 ft. and contains an adjustable Prandtl Pitot tube immediately ahead of the entrance to the exhaust fan. The fan *H* is connected to the duct section *G* by means of a flat plate with a circular opening equal in diameter to the front of the fan inlet piece. The fan is directly connected to the driving motor, which is mounted in a cradle dynamometer.

Whereas Fig. 1 shows the duct with its inlet at the ground line and otherwise unobstructed, the actual inlet conditions are illustrated by Fig. 2. Although the inlet is about 6 in. above the ground level, both sides and the top of the inlet are surrounded by the walls and roof of an auxiliary building. The original purpose of this building was to permit recirculation of air from the main shop in order to maintain comfortable winter temperature in the discharge house shown in Fig. 3. This objective was abandoned, however, when the uniformity of duct flow was found to be impaired. The end wall of the auxiliary house opposite the duct inlet was therefore removed.

The existing conditions at the duct inlet, although not ideal, permit fairly uniform and steady flow maintenance at section *C*, where the volumes are measured. Undoubtedly the "egg-crate" straightener at the front of section *B* (see Fig. 4) appreciably contributes to the maintenance of flow distribution shown by Fig. 5. What is more important is the fact that the flow pattern remains practically constant both for

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Fig. 2.—Inlet to duct, showing instrument house and discharge house in background.
Fig. 3.—Discharge house, showing auxiliary fan.
Fig. 4.—Duct inlet, showing "egg-crate" straightener section and shutters.

different settings of the shutters and for different volumes at fixed shutter setting accomplished by varying the speed of the fan. Figs. 6 and 7 show the constancy of the relationship existing between the air velocity at the center of the duct and the mean velocity weighted with respect to duct area, obtained by 20-position Pitot-tube traverses.

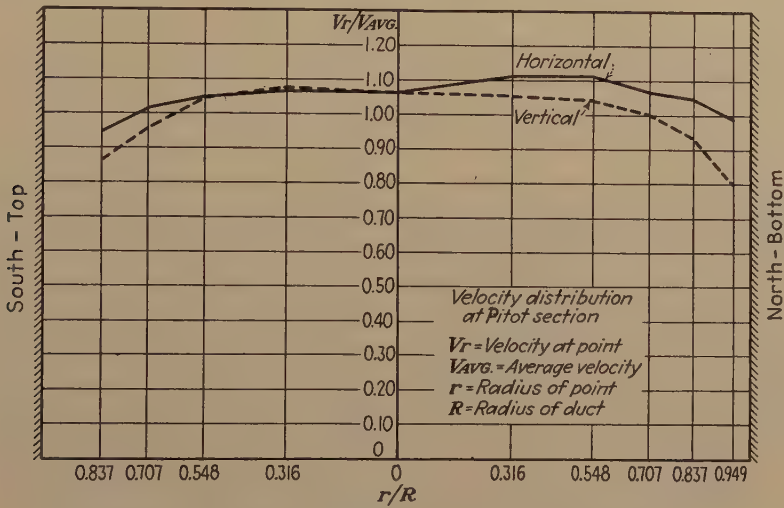

FIG. 5.—VELOCITY DISTRIBUTION AT PITOT SECTION C.

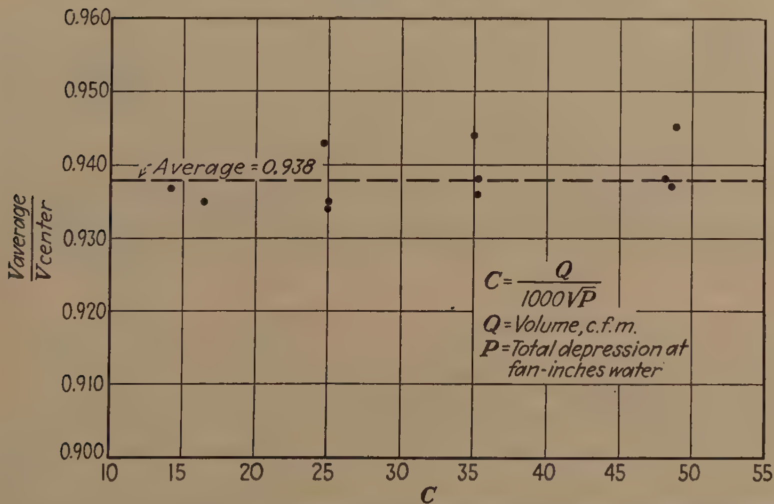

FIG. 6.—DUCT CALIBRATION. CENTER CONSTANT WITH VARYING SHUTTER SETTING.

Duct sections *B* and *C* are selected of sufficiently small cross-sectional area to establish air velocities ranging from 2000 to 6000 ft. per minute, thereby permitting accurate Pitot-tube velocity traverses. Air velocities, and consequently air volumes, can be determined more accurately from

proper Pitot-tube application than is possible with an anemometer, velometer or any similar device. The most accurate Pitot-tube velocity measurements can be obtained, however, in velocities of 2000 ft. per minute or over, which is one reason why the Pitot tube is discarded in favor of the anemometer when measuring low air velocities found in mine airways. However, an accurate correction factor for duct center velocity having been established, the air volume can be determined instantaneously without the necessity for maintaining constant fan speed during the time interval required for a complete section traverse. This permits both greater speed and greater accuracy. Section C is far enough in front of section E to be free from any influence that the shutters may have upon upstream air.

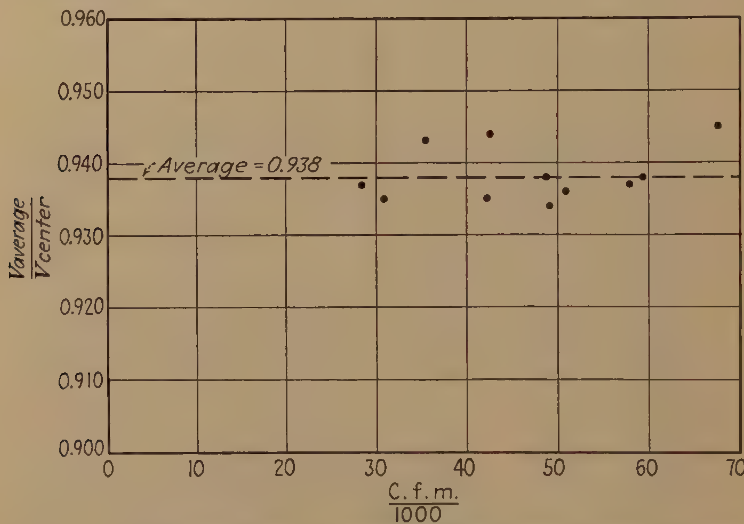

FIG. 7.—DUCT CALIBRATION. CENTER CONSTANT WITH VARYING VOLUME.

The pressure in front of the fan is determined by measuring the static depression with the centrally located Pitot tube at section G. Careful traversing shows that the static pressure over the entire section area is sufficiently constant to permit the use of the static depression measured at the center of the section. The velocity pressure at this section is calculated from the mean axial velocity as determined by dividing the air volume per unit time by the section area. The total pressure at the fan inlet is therefore considered as the algebraic sum of the static depression and the velocity pressure. In order to measure the total depression directly it would be necessary to traverse the complete section in front of the fan, because of the extent to which the total pressure distribution varies across the section. It is of decided advantage to obtain an instantaneous pressure reading considering both time and the elimination of necessary fan-speed control.

With the fan exhausting into the atmosphere, the total pressure on the inlet side of the fan represents the static pressure across the fan.

Fig. 3 is a view of actual conditions at the discharge end of the duct. The exhaust fan being tested discharges into a building equipped with an auxiliary propeller fan mounted in the far building wall, as shown. The purpose of the auxiliary fan is to maintain a subatmospheric depression in the discharge house sufficient to permit the testing of the fan to the point of free delivery. Consequently, the fan's static pressure becomes the difference between the total pressure at the fan inlet and the static pressure in the discharge house.

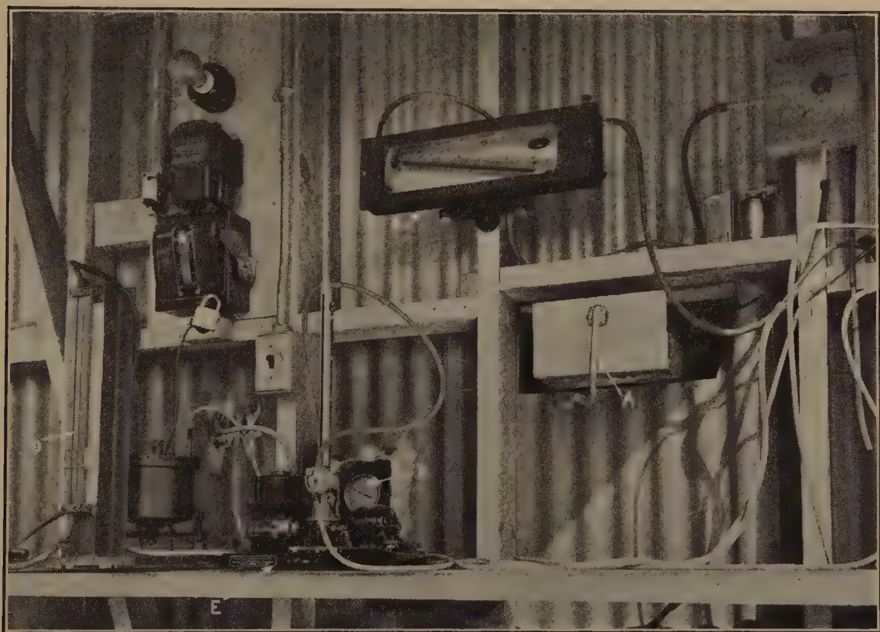

FIG. 8.—INTERIOR OF INSTRUMENT HOUSE.

Fig. 8 shows the instrument house in which are: *A*, the crank controlling duct resistance shutters; *B*, a thermometer indicating the dry-bulb temperature at section *C*; *C*, a barometer; *D*, a vertical manometer equipped with double slide rule glass for measuring the differential static pressure across the test fan; *E*, a micromanometer designed by the National Advisory Committee for Aeronautics, used to accurately measure velocity pressure at section *C* and particularly well suited for such measurements in the presence of fluctuating flow; *F*, an inclined manometer used with two-way cock to record static pressure reductions at the base of the test-fan discharge diffuser.

The duties of the observer stationed in this instrument house are the following: (1) control of the duct resistance; (2) recording the dry-bulb

temperature of the air at section C; (3) recording the barometer at the beginning and end of each test; (4) recording the static depression in duct section G near the fan inlet; (5) recording the velocity pressure at the center of section C; (6) recording static depressions at the base of the test-fan diffuser.

In the discharge house (Fig. 3) are the dynamometer scale, an electric tachometer indicating the speed of the test fan and wet-bulb and dry-bulb thermometers measuring these temperatures in the air entering the fan inlet. The observer stationed in the discharge house has the duties of: (1) recording the dynamometer scale reading; (2) recording the speed of the fan; (3) recording the wet-bulb and dry-bulb temperatures of the air at the fan inlet.

There are signal lights in both the instrument house and the discharge house, each operated by a switch in the other house. The operator in the instrument house first adjusts the resistance shutters to a desired setting, then turns on the signal light in the discharge house, thereby signifying his readiness to start reading the instruments. When the operator at the discharge house is ready to proceed with the readings, he turns on the light in the instrument house and then pays close attention to the readings of the dynamometer scale and the electric tachometer. In the meantime the instrument-house operator records the static depression in duct section H, the static depressions at the base of the test-fan diffuser, the center velocity pressure and the dry-bulb thermometer at duct section C. Ordinarily the instrument-house operator requires more time for his readings than does the discharge-house operator. Therefore, the instrument-house operator, when his readings are complete, signals the discharge-house operator by turning off the signal light. If, as usually happens, the discharge-house operator has completed his readings, he so signifies by turning off the instrument-house light. The instrument-house operator then changes the setting of the resistance shutters, during which time the discharge-house operator records the wet and dry-bulb temperatures of the air entering the fan inlet. This cycle is then repeated for the new setting of the shutters.

Complete readings for 25 settings of the resistance shutters can be made in approximately 90 min. of actual testing time. This speed is possible only because instantaneous volume readings can be made with a calibrated duct. This is of particular value when testing propeller fans equipped with adjustable rotor blades and adjustable guide vanes. With seven adjustments of the former and five of the latter it is necessary to run 35 complete performance curves when testing one fan.

At the beginning of each test the following precautionary measures are a matter of routine:

1. All hose connections are tested for leakage by disconnecting at the connection farthest from the manometer, inflating and ascertaining

that a state of equilibrium is maintained by the manometer liquid column.

2. Exact multiplier constants are determined for each manometer by comparing with a vertical manometer containing water and graduated in inches, thus accounting for changes in specific gravity of the various manometer fluids with changes in temperature.

3. Both the test fan and auxiliary fan are given a preliminary "warm-up" run of 30 min., following which the tare reading on the dynamometer scale is recorded with the test-fan motor running light. This tare reading is repeated at the end of the test.

The consistency of data thus acquired is attested by Fig. 9, showing pressures and powers as ordinates plotted against volumes as abscissas.

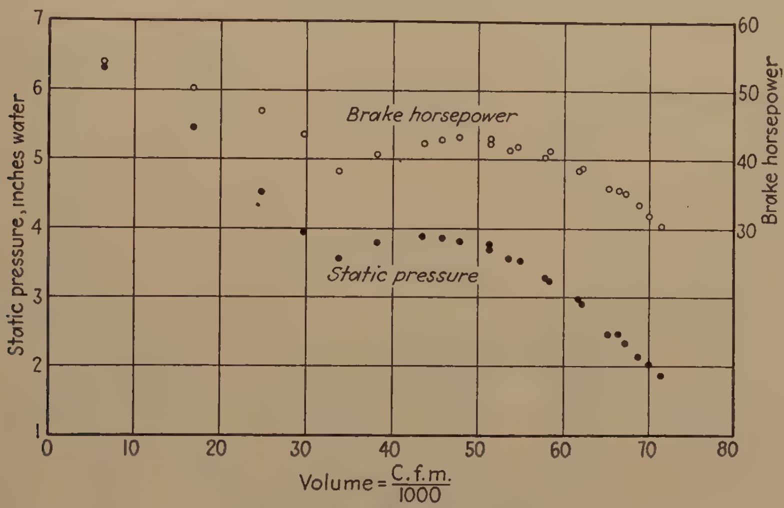

Fig. 9.—Specimen fan test.

Such consistency is possible only because the test setup permits practically instantaneous and simultaneous recording of the essential data. Eliminating the necessity of maintaining constant fan speed under varying voltages while conducting a complete velocity traverse rules out the following customary sources of error: (1) volume and pressure discrepancies due to lags in speed correction; (2) wide variations of the dynamometer scale indicator resulting from the reactive torques accompanying speed corrections.

In spite of the many and elaborate precautionary measures employed and the apparently accurate and consistent test data available with the existing equipment, there is room for additional refinements, which we would incorporate in any test duct to be built in the future. There is excessive turbulence in the expansion section F resulting from one or a combination of the following factors: (1) too rapid expansion with rapidly

increasing and unstable boundary layers; (2) corner effects; (3) location of resistance vanes at entrance to section *F*.

The result is a gusty nonuniform flow condition set up throughout section *G*, which is materially influenced by different settings of the resistance shutters. It is possible to establish actual flow reversal in the corners of section *G* for certain settings of the resistance shutters. This effect is apparently not propagated to the higher-velocity air entering the blades of the test fan, judging from Pitot-tube traverses immediately ahead of the fan rotor. However, flow conditions in section *G* might be materially improved by one of the following changes, or a combination of them: (1) lengthening section *F*; (2) rounding the corners of section *F*; (3) maintaining a duct of circular section throughout; (4) relocating resistance shutters at discharge end of section *F*.

The most ideal flow conditions could be established throughout the duct by mounting the entire duct on stilts so that the duct inlet might remain unobstructed in all directions by a distance equal to several inlet diameters. Unfortunately, such an installation would be too costly to be practical.

An interesting effect produced by the turbulent flow in section *F* is the consistent measuring of 5 per cent more air entering the test fan than passes section *C*. This discrepancy occurs with Pitot-tube traverses taken immediately ahead of the test-fan rotor when operating with the auxiliary fan idle. The same is true for Pitot-tube traverses taken in the test-fan casing in the plane of the fan straightener vanes when both rotor blades and straightener vanes are removed and air is drawn through the duct and test fan by operating the auxiliary fan alone. The only places where leakage can occur are: (1) at section *E*, where the resistance-shutter controls project out through the duct; (2) at a slot in the top of section *G*, which is available for traversing this duct section with an anemometer operated from outside the duct, but which is closed off and sealed with tar; (3) an entrance door bolted to the side of section *G* to facilitate easy access to and from the duct; (4) the various joints of the duct, which are sealed with tar.

The amount of excess air, however, appeared too large to be accounted for by leakage alone. Furthermore, the percentage of leakage was independent of the static depression maintained in duct section *G*. In order to determine whether or not leakage was present to any such extent as suggested, the duct was sealed off at section *C* by bolting a solid flat plate immediately ahead of the section. A similar plate was bolted across the end of section *G* immediately adjacent to the test fan. A 16-in. blower was installed within the duct, blowing through a small pipe passing through the sealing plate across section *G*. When the blower was operated a static depression of 10.75 in. water gauge was produced within the duct, which produced a total leakage of approxi-

mately 245 cu. ft. of air per minute. This test proved that leakage did not account for the volume difference.

The accuracy of Pitot-tube readings taken in front of the test-fan rotor might be questioned because of possible pulsations transmitted to the air by the fan blades. The frequency of such pulsations would be the product of the rate of rotor rotation and the number of rotor blades.

This type of high-frequency pulsation cannot be detected by watching the manometer liquid column. However, the manometer reads too high. The Pitot-tube manometer reads a value proportional to the mean of the squares of the instantaneous velocities. This value exceeds the square of the mean velocity and so accounts for the inaccuracy of volumes so computed. The amount of error depends upon the amplitude or range through which the instantaneous velocities vary; also upon the manner in which these velocities vary with respect to time.

In order to account for a 5 per cent volume error if the velocity change is linear, the instantaneous velocities must vary between the limits of 45 and 155 per cent the mean velocity. If the velocity varies sinusoidally with respect to time, it is necessary that the range extend from 55 to 145 per cent the mean velocity in order to account for a 5 per cent error in volume.

The likelihood of this type of pulsation occurring ahead of the fan wheel is remote, even though there may be such a tendency behind the rotor. Since the same volume discrepancy appears in the cast of Pitot-tube traverses taken within the test-fan casing with the auxiliary fan alone in operation, there is little likelihood that this type of high-frequency pulsation accounts for the apparent difference in volumes.

The apparently high volume readings obtained at the test fan are, however, undoubtedly the result of a form of high-frequency pulsation in the air entering the fan. This pulsation is probably the result of the imperfect expansion accomplished in section F.

Retarded flow in this section produces eddy currents superimposed upon the main air stream, which are caused by back flow in the boundary layers. The Pitot tube reads high because of measuring the velocity variations resulting from these swirls, or so-called eddy currents. If this explanation is correct it then does not follow that the entire body of air within the duct undergoes velocity changes of the magnitude required to produce a volume reading 5 per cent too high.

These is a very definite vibration felt in the walls of the steel duct, which obviously is not transmitted to the duct by the test fan, which itself is independently mounted on concrete foundations. These vibrations are of very high frequency and undoubtedly are the sensible manifestation of the turbulence produced in expansion section F.

There may still be some question as to the effect of this turbulence on the actual performance of the fan being tested. As has already been

pointed out, the velocity distribution within the fan inlet is good. However, the presence of the suspected swirls might have a detrimental effect upon the fan's performance. Unfortunately these swirls do not produce any fixed direction of flow, since they may be pictured as being carried along with the main air current. The only evidence of their presence detected by measurement is the apparently high volume reading at the fan inlet.

Experimental wind-tunnel tests indicate that the lift and drag characteristics of an airfoil section are not materially affected by the presence of turbulence. Therefore it may be assumed that there is small adverse effect upon the fan's performance by the presence of turbulent air in section *G*.

Although the test equipment described herein is not perfect, it offers a practical means of testing propeller mine fans in the factory with a higher degree of accuracy than has heretofore been attainable. Whereas the air enters the fan axially from the test duct, an endeavor is made to attain a similar state at the mine by locating air-turning vanes at the top of the airshaft.

The Jeffrey Manufacturing Co. will furnish to any interested fan manufacturer drawings or other available information pertaining to this test procedure. The mining industry could further its own interest by requiring all manufacturers to present test data obtained by a standard test procedure. This would remove many mysteries surrounding mine-fan application and eliminate evils that have resulted from the use of different and inadequate test methods at the factory.

DISCUSSION

(*C. T. Hayden presiding*)

C. Lee,* Chicago, Ill.—Mr. Mancha's paper is a frank and valuable discussion of one of the problems of the large fan manufacturer. Few operating companies have the facilities or the personnel to check the performance of fans in the field. Seldom even does a representative of the operating company witness a factory test of a fan. Such a test is technical and preparations may be tedious.

In this paper very little reference is made to the dollars and cents value of high-efficiency fans to the operating company. Assume a mine requiring 100,000 cu. ft. per min. at 4-in. water gauge with 88 per cent efficiency motor and purchased power at 1½¢ per kilowatt-hour. The annual costs would then be affected by the fan efficiency somewhat as shown in the table on p. 121.

Since the mine fan usually runs continuously, whether the plant produces or not, the cost of the fan operation is very much the same as a fixed charge due to capital investment. Thus it is apparent that high-efficiency fans are important to the operator that expects to meet keen competition.

It would appear to be desirable that all manufacturers agree on and adopt standards of testing to include all pertinent details, so that the customer could rely on the data for comparative purposes.

* Peabody Coal Co.

The discrepancy mentioned in the paper—5 per cent more air entering the fan than is measured at the point C—should warrant further checking. The customer would wish to know which value is used.

Fan Efficiency	Annual Cost	Difference
30	$23,300	
40	17,500	5,800
50	14,000	3,500
60	11,650	2,350
70	10,000	1,650
80	8,760	1,240

This test setup takes the air in through a circular tube, then through regulating valves, thence into a square section, thence back into a circular section. It seems that the entire channel should be circular for this type of fan. There appears to be one particularly bad feature about the tunnel; that is where the square section G connects to the fan inlet. If it is understood correctly, there are four flat surfaces projected into the air stream where the inscribed circle of the fan is connected into the square box.

Elimination of the shutter section should reduce the eddy currents, like smoke rings, from the expansion chamber. As a substitute for this shutter it might be possible to use multiple layers of filter mats, say of spun glass over the intake A. If so used, the present 3-in. tubes to straighten flow might not be necessary, as the turbulence of passing through the filter would be fairly small as compared to that caused by a shutter or regulator.

One question might be raised, referring to the constants and calibration of Pitot tubes. It is possible that the empirical constants have not been accurately determined for various shapes of tubes and for all velocities of flow.

In regard to the harmonic waves or pulsation affecting the Pitot tubes, it might be possible with modern electrical and acoustical pickup devices that such disturbance could be amplified and projected on a cathode-ray oscillograph in order to investigate the magnitude and frequency of such harmonics.

Despite the few discrepancies mentioned by the author, the paper is a timely and valuable contribution on the subject.

Safety Practices of the Koppers Coal Company

By L. C. Campbell,* Member A.I.M.E.

(Chicago Meeting, October 1938)

The purpose of any accident-prevention program is the curtailment or entire elimination of injuries and fatalities. It is a job that is never finished in the coal-mining industry. Day by day, on shift and off shift, men are exposed to injury and death in varying numbers. Seldom is it possible to lock up at the end of the shift and leave a mine plant in the care of a sleepy watchman, as can be done at most manufacturing plants. Even on off shifts there are maintenance and supply crews, coal cutters, pumpers and ventilation men exposed to danger.

The prevention program of The Koppers Coal Co. recognizes two primary facts:

First, no program can be successful without the full support of the management and complete knowledge of the job at hand on the part of every supervisor.

Second, and equally important, is the full cooperation of every employee in observing the safety rules and practices set up by the management. Conscious day-to-day concern for his safety by the individual worker is the longest stride toward a good safety experience.

How to accomplish the education of the individual worker and supervisor for safety is the problem of The Koppers Coal Co., as it is the problem of all coal companies.

Safety Rules

A simply stated set of Safety Rules and Practices for the government of employees is the starting point with The Koppers Coal Co. There are but 13 of these rules, which are short, easily understood and followed by a campaign of education and inspection. Each employee receives a set of these Safety Rules and Practices and signs a receipt for them, which is bound in at the back of the booklet, and this receipt becomes a part of his employment record. These rules and practices are as follows:

Introduction

The following rules have been compiled for the safety of employees and are established according to rules number one and number two of the official bulletin

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* Assistant to the Vice President, The Koppers Coal Co., Pittsburgh, Pa.

published by the Compensation Commissioner of the State of West Virginia under date of February fifteenth, nineteen hundred and twenty nine. The failure of any employee to abide by these rules, thereby endangering his own person, or the person or lives of his fellow employees, shall be sufficient cause for dismissal and the refusal of award of compensation for any injury to himself resulting from such violation.

The intent of these rules is to aid in the enforcement of the state mining laws and in no way to conflict with same.

Rules for Employees

Rule 1.—It shall be the duty of every employee of this company to fully comply with the Mining Laws of the State, and to observe every precaution to prevent accidents; and to fully comply with and obey orders of the section foreman, mine foreman or their superior officers.

Rule 2.—All miners shall have and keep in proper condition a full set of tools, in accordance with the list posted at the mine foreman's office.

Rule 3.—All employees upon entering the mine shall go direct to their working places and no person shall be permitted to loiter in or about the mine, buildings or machinery, or go into an abandoned part of the mine without written permission from the mine foreman. Every employee is forbidden from tampering with any door used for ventilation, or machinery or equipment in or about the mines except in discharge of his duty.

Rule 4.—On entering his working place, each miner shall make a careful examination of same, and take down all dangerous slate, with a slate bar, or make it safe by proper timbering. Safety posts shall be set in accordance with instructions before starting work.

Whenever slate of any kind is removed safety timbers must be set to prevent the falling of slate over or nearby.

Rule 5.—No employee shall take into the mine a larger quantity of explosives than may be required for a shift (and at no time shall more than 5 pounds of explosives and more than 7 detonators be kept inside the mine by any employee).

All explosives hauled into or out of the mine shall be in compliance with the State Mining Laws.

Explosives and detonators must be carried and kept in a safe receptacle approved by the Superintendent and must be stored separately in cubby holes in the rib at least ten feet apart, and at least seventy-five feet from the working face.

No charge of any explosives in any one coal shot shall exceed one and one half ($1\frac{1}{2}$) pounds and no hole shall be charged with more than one kind of explosive, and not more than one shot shall be fired at a time.

All holes must be tamped with clay to the mouth of the holes, using wooden sticks for tamping.

All shots shall be fired by electric detonators and approved battery only. No shooting shall be done off power lines.

No shots shall be fired in any place known to liberate explosive gas until such place has been properly examined by an authorized and competent person and approved for work. No shot firing cable less than 100 feet long will be permitted. All employees must remain in a sheltered place at least 100 feet away from where shot is being fired.

Rule 6.—After a shot has been fired the employee before starting work shall make a careful examination as to the condition of the roof and its safety. In the case of misfires the Assistant Foreman shall be promptly notified and NO PERSON shall return to working place where misfire occurred until given permission to do so by the Assistant Foreman, who will personally direct drilling and charging another hole.

Rule 7.—All persons are forbidden to ride upon any incline or upon any car, conveyor lines, engine, motor or other contrivance except as permitted by law. No employee shall travel to or from his work upon any slope, plane or motor road when another road is provided. When necessary to travel upon any such slope, plane, road or haulway, employee must use every precaution to prevent accident to himself and others.

Rule 8.—All cutting machine operators must carefully examine the place they are to cut for loose slate or other unsafe conditions and must not cut such a place until it is made safe. In mines where gas is likely to be encountered, they must test for gas with a safety lamp and if the presence of gas is detected, they must not operate machine until gas is removed and the place is pronounced safe. In any gaseous mine no cutting machine shall be operated more than thirty minutes without examination for the presence of gas. If any is found, the operator shall immediately stop the machine and cut off the current, and machine shall not be started until gas is removed and place pronounced safe. Cutting machine operators must not permit any visiting in any place in which machine is being operated.

Rule 9.—All employees are prohibited from wearing loose or hanging clothing or apparel which may endanger life or limb by being caught in moving machinery, including power drills.

Rule 10.—Motormen and brakemen shall not permit any person to ride on any locomotive or trip, except those duly authorized and must stop trip to enforce this rule.

Motormen shall not operate any locomotive without burning headlights, or with defective brakes, or sand rigging.

Motormen shall sound an alarm on approaching curves, doors, check curtains, cross entries and places where workmen are near the track.

Motormen, brakemen or other authorized persons are prohibited from operating locomotive with pole in reverse position or cleaning sand pipe or sanding rails by hand while locomotive is in motion.

When hauling rails, pipe and other materials, an empty car shall be placed between the locomotive and the material truck.

No motorman shall get off his motor, without closing his controller.

Jumping off or on the front end of a moving trip for any purpose is prohibited. Cars shall not be coupled or uncoupled while in motion and coupling hooks shall be used at all times while making couplings.

No one is permitted to ride between cars on any trips whatsoever.

Rule 11.—Every miner shall adequately block his car with a clevis or similar safety device which will be furnished by the company, and brakemen shall use clevises or safety chains to block standing loads or empties on sidetracks. No wedges, chips of wood, et cetera, shall be used at any time to secure standing cars.

Rule 12.—All employes are required to wear goggles of an approved type when working with pointed or edged tools, and when engaged in sledging rock, driving spikes, using grinding wheels, or any other occupation where there are flying particles.

Rule 13.—All sprockets, gears and coupling drives shall be guarded at all times. In conveyor mining, a safety zone at least six feet from the tail sprocket shall be maintained for the safe removal of timber and other supplies. An adequate signalling system shall be maintained between the working faces and the loading point.

Inspection and Records Data

The Accident Prevention Department works under the direction of the Operating Vice President. Until recently, the department was made up of an engineer in charge at Pittsburgh, a general inspector, centrally

located in the field, and four division inspectors. The division inspectors now report directly to the division superintendents, and the general inspector to the Operating Vice President. The change is in the interest of bringing responsibility for accident prevention closer to the operating heads. It is hoped thereby to place it at least on an equal footing with cost, production, preparation, etc.

There is a local inspector at every plant, who is under the direction of the plant superintendent and also works with the division safety inspector. The division inspectors spend most of their time in the mines making inspections of sections in accordance with a standard rating sheet on which specific charges are made for substandard conditions based upon the safety rules and practices. For example, substandard timbering is charged with 200 points, brows or overhanging ribs 40 points, etc. All charges are then added and a percentage of possible total charges computed. This is termed the "rating" for that section. If the rating is over 80 per cent, it is considered satisfactory, but if it is under 80 per cent the foreman is cautioned by the superintendent, who receives a copy of the rating sheet as soon as the inspector comes out of the mine. A series of ratings under 80 per cent indicates that the foreman's work is unsatisfactory and results in his replacement. Since each check number is shown on the rating sheet, it is possible for the foreman and superintendent to learn which individual workers are responsible for the substandard conditions found. A copy of this rating report is sent to the Pittsburgh office, where a complete record of each foreman is kept. It shows the number of accidents charged against him, the number of man-hours worked each month under his supervision, and a record of the rating he made.

Accident Reports

Each accident is reported to the superintendent, division superintendent, safety inspector and engineer in charge of accident prevention. Serious accidents are promptly investigated by the local or division inspector, and fatal accidents are investigated by the general inspector, who procures statements from all witnesses and compiles a report with recommendations.

The names of foremen who have gone a year or more without a lost-time accident are printed each month on an Honor Roll, showing the total time worked without an accident. It is posted conspicuously at each plant.

When the name of a foreman first appears on the Honor Roll, a letter of congratulation is sent to him by the Vice President. A similar letter is sent at the end of each 12 months to all foremen whose names continue on the Honor Roll. This roll now contains the names of 147 foremen

It has increased steadily from 40, in March 1933, when the Honor Roll was started.

First-aid Training and Safety Accessories

First-aid training plays an important part in the safety program, practically all mines being 100 per cent first-aid trained. For the past four years teams from The Koppers Coal Co. have won first place for both white and colored teams in the annual contests, and first place for white teams in another of the yearly meets held by the West Virginia Department of Mines. With one exception, all mines are fully equipped with hard-toe shoes and hard hats, with a resulting decrease in foot and head injuries. The use of goggles is almost universal.

There are bulletin boards at each plant on which safety posters of the Elliott Service Co. and National Safety Council are posted weekly. Several Safety Dramalogues, with traveling belts carrying a safety message, are also used. Recently moving pictures on safety, taken at some of the plants with employees appearing in the several parts, have been used. These are stories covering actual accidents in connection with haulage, timbering, etc. They have been shown in local theaters, schools and safety meetings to several thousand employees and their families. We believe these pictures to be one of the most effective forms of accident prevention for plant and community appeal.

Meetings of foremen are held frequently at all plants. Safety measures are discussed and the recent accidents studied, with a view to preventing a recurrence. At most plants, monthly safety meetings are held and recommendations from workers on how to prevent accidents are invited. Many helpful suggestions are received.

Discipline

Discipline must of necessity play a large part in enforcing standards and safety practices. Conditions are so different in most of the mines that each has its standard timbering plan. All other safety rules and practices are as uniform as it is possible to make them at every plant. Standardized discipline for the violation of safety rules including timbering consists of suspension of one day for the first offense, three days for the second offense and discharge for the third offense in a 30-day period. A careful record of all discipline by plant and employee name and check number is kept. Many workers that are not safety-conscious are singled out from this discipline record.

Results

The safety program at the mines of The Koppers Coal Co. has been conducted under a severe handicap. With two exceptions the properties have been acquired when in a badly run-down condition, and it has

taken years of work and the expenditure of large sums of money to bring the properties into such physical condition that the management could feel it had done its part toward safety and could properly demand that the employees do their part. The mines are widely scattered in Kentucky, West Virginia and Pennsylvania. Production is more than 1,000,000 tons monthly from seams that range from less than 3 ft. to more than 8 ft. thick. The 27 mines are opened either as shafts, drifts or slopes. They present every hazard known to bituminous coal mining.

The results of this program have been progressively encouraging, and the steady reduction in compensation rates has fully demonstrated the economic value of the safety work. The present rate in West Virginia for Koppers mines is $2.54 per hundred dollars of payroll. Only two of the larger operating companies, and these have a production equal to but one of the Koppers divisions, have better rates.

The Sentinels of Safety trophy was won by Koppers mines in 1936 and again in 1937. In each year the award was to mines that had no lost-time accidents whatever for the entire year. This year (1938) a certificate was awarded by the Joseph A. Holmes Safety Association for having 184 employees who have worked in and around coal mines for 20 years or more without a lost-time accident. Of these, 14 have worked over 50 years without such an accident.

We acknowledge the fine support and assistance of the several State Departments of Mines, and of the United States Bureau of Mines in helping to point the way to safety. Mr. T. E. Lightfoot, Engineer in Charge of Accident Prevention and Compensation, deserves a large measure of commendation for the tireless way in which he has worked for safety. Today the most serious problem we face is to improve a bad fatality record, which has held on regardless of fine improvement in every other record. We are carrying this fight to the individual worker and supervisor, with a firm conviction that with their help the fatality figures can be halved in 1939.

DISCUSSION

(*Gordon MacVean presiding*)

L. C. Campbell, in reply to a question by Willard Gibbs,* of Pittsburgh, Pa., stated that rock dusting is a major safety practice; 65 per cent incombustible is set as the lower limit, and reports of rock-dust samples are required each week. The samples are collected from both the roofs and bottoms and are analyzed in a laboratory.

A. J. Bartlett,† Holden, W. Va.—Safety practices of the Koppers Coal Co. are similar to those of other organized accident-prevention programs. Whole volumes could be written on the subject and still many questions unanswered as to what method produces the best results. The company should be commended for the considerable progress it has made in this work.

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* Harwick Coal and Coke Co.

† Chief Mine Inspector, Island Creek Coal Co.

All mining properties, regardless of their natural conditions, have inherent hazards, which can be reduced to a minimum by the application of the proper methods. The greatest hazard in any mine is the human element or mental attitude of the worker, that natural instinct "to take a chance," thus the necessity for safety rules and regulations. The Koppers Coal Co., one of the largest producers in the coal-mining industry, believes in a short set of safety rules. It is established as a fact, the fewer and shorter the safety rules, the more readily they are understood and practiced by the employees.

Before selling safety to the employee, the management must set its own house in order and must demonstrate that it really means business in its safety program, by first providing a safe mine, safe equipment, safe methods and trained foremen. Too much emphasis cannot be placed upon the fact that upon the management rests the success or failure of any accident-prevention program. Theirs is the responsibility for the safety of the entire organization. If they are personally sold on safety it will be reflected down through the organization to the rank and file.

Cooperation and confidence are very necessary factors in accident-prevention work. The management must have the cooperation of its employees and the employees must have confidence in the management and its policies. These two factors will bring the desired results.

A successful accident-prevention program cannot be derived from a few bursts of enthusiasm, and then forgotten; it must be an everlasting job, day after day, hour after hour; employees must be made safety minded by education, they must be constantly reminded of the right and safe way to do their work. Personal contact of employees at their respective work, by inspectors, safety directors, etc., will bring good results.

A successful safety program necessitates the keeping of accident records, compiling accident statistics and analyzing all accidents. These data should be presented in an understanding manner to those in supervisory positions, and have been found effective aids in preventing accidents.

By keeping accurate records, a constant check is maintained upon the results of the safety program. Just as monthly reports are given to the management regarding operating cost, so should the safety record for the month be considered an item of equal concern. In this way the management can see what progress is being made and where failures are appearing in the safety program.

Every accident should be used as a means of preventing recurrence of similar accidents, by making a careful study of its causes, placing responsibility where it rightly belongs, and recommending changes in the physical conditions and procedures that contributed to the accident. However, all of this effort will be lost unless the recommendations are carried out.

Recognition should be given by the management to all employees and supervisors that maintain good accident records, and those with bad accident records should be reprimanded.

J. V. Berry,* Johnstown, Pa.—The dangers in coal mines not only during the work shift but continuing during the off shift are not recognized by many, especially when comparing coal-mine accidents with those of other industries. The adoption of a few safety rules and practices for the guidance of employees, and a campaign of education so that all will understand them, follow them, and assist in their enforcement, is a very important part of a safety program. Some companies adopt so many rules and regulations that neither the officials nor the employees know them, with the result that few are followed. The adoption of any rule or rules that cannot be

* Supervisor of Safety, Industrial Collieries Corporation.

followed and enforced causes disrespect for all others. Safety rules to be effective should be as few as possible, known by the officials and employees, and rigidly enforced.

This company's setup for its inspection force, the local and division inspectors being responsible to the plant superintendent, should bring good results; and the rating of sections and mines will induce each official to strive for a good rating. I was particularly interested in the inspection of the outside of the mines. Many companies fail to realize that no inspection is complete unless it includes the outside operations.

Investigation of all accidents with the intention of finding the cause and putting into operation methods of preventing recurrences is bound to have its effect. Each foreman will strive to keep his record clear so as to have his name on the honor roll; and one thing every workman values is recognition of his efforts, especially from the vice-president of his company. This is evident from the record of this company, and from the fact that its honor roll contains the names of 147 foremen.

As Mr. Campbell points out, 100 per cent first-aid training, the use of hard-toe shoes, hard hats, safety goggles, and the dissemination of safety messages on well-kept bulletin boards, and the use of motion pictures that tell the story of actual accidents at their own mines, all play an important part in a safety program. Another important phase of this program, as I see it, is the frequent meetings held by the officials at the various plants, during which all accidents are discussed with the view of preventing recurrences, and suggestions are received from the employees on accident prevention. As pointed out, discipline plays an important part, and I believe that in most mines that have a high accident rate there is poor discipline.

I congratulate this company on having shown, even under adverse conditions, a steady progress toward accident prevention.

Organized Safety in the Anthracite Mines of the Susquehanna Collieries Company

By C. G. Brehm*

(Chicago Meeting, October 1938)

The anthracite-producing region is in the northeastern section of Pennsylvania, and has an area of approximately 484 square miles. It is divided geographically into three separate fields, known as the Northern, Middle and Southern.

The Northern field, with Wilkes-Barre approximately at its center, is a basin about 60 miles long and almost 6 miles wide at its greatest distance across. Generally speaking, it may be said that the coal measures in this field lie horizontally, or on slight pitches, and consist of from 11 to 19 veins of coal, varying in thickness from several inches to several feet. The intervening strata between veins may be composed of slate, clay, rock or shale, or a combination of several, and are sometimes very thin. This condition often causes a treacherous roof, therefore roof support and the attending hazard of mining are problems of concern.

The Middle field is made up of a number of small basins of not very great depth, and the measures, which are in like number to those of the Northern field, are varying inclinations running from nearly flat to steep pitches.

The Lower, or Southern field, is made up of basins of great depth, some being 4000 ft. deep, and its measures are mostly on very steep pitches. In certain sections this Southern field is being mined at an elevation of 1300 ft. below sea level, on a pitch ranging from 50° to practically vertical, and because of the tremendous pressure and the inherent gas in the seams, outbursts of coal frequently occur at the face, locally known as "bumps." These outbursts are accompanied by a rumbling noise, after which gas is usually found. This condition makes mining a difficult and extremely hazardous undertaking, and unusual skill is required on the part of the miner to drive and maintain openings necessary to the production of coal. Miners working in such seams on the heavy pitch must carry face batteries (Fig. 1) to afford proper protection while working at the face, because of the outbursts and free running coal. To drive haulageways or gangways in this field it is

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usually necessary to use heavy timber, the top and sides must be lagged, and forepoles used in advancing the face to prevent the coal from running away and filling the gangway.

Throughout the entire anthracite region, in many of the veins second and even third mining is being conducted, with its attending high costs and hazards. Some of the veins, particularly the thin veins of the flat measures, are mined mechanically. In such a field it is natural that underground development is most extensive and transportation systems and methods are required almost equal to those of a modern railroad.

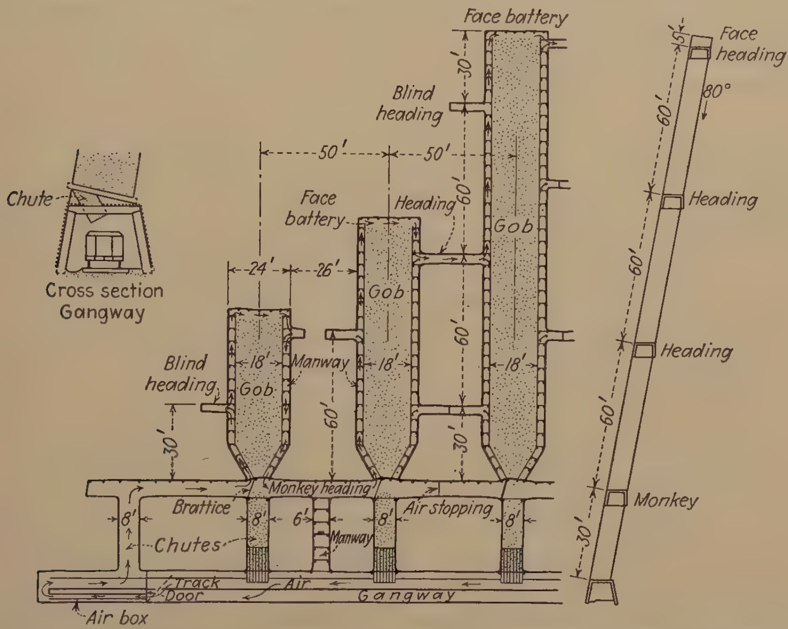

FIG. 1.—MINING PITCHING SEAMS, WILLIAMSTOWN COLLIERY, SUSQUEHANNA COLLIERIES COMPANY.

Because of the hardness of the coal and the percentage of rock work, great quantities of explosives are used. In the entire anthracite field, more than 25,000,000 lb. of explosives is used in a single year. The figures for the year 1936, according to reports of the Pennsylvania Department of Mines, are as follows: black powder, 6,252,300 lb.; dynamite, 12,449,516; permissible explosives, 13,362,815; total, 32,064,631.

Vast quantities of timber are necessary for roof support, and many millions of gallons of water are pumped annually.

Many of the veins generate an explosive gas, therefore proper ventilation is a matter of much importance, entailing not only large ventilating fans on the surface but miles of airways within the mines.

This, rather briefly, gives some idea of the anthracite coal fields of Pennsylvania, the vast underground workings, the great percentage of

rock and refuse to be handled per ton of marketable coal produced, difficulty of mining, ventilation and drainage, and the attending hazards of such an industry.

The companies of this region range in size from those operating a single mine to the larger companies operating a number of mines, and the safety organizations of these various companies range from the simplest type to the more elaborate. Many of these companies have efficient safety organizations. The following detailed plan of the Susquehanna Collieries Co. is indicative of the safety organizations of the larger companies, many of whom have either adopted the same plan or some modification of it.

Susquehanna Collieries Company

The Susquehanna Collieries Co., operating mines in the Northern, Middle and Southern fields, has been vitally interested in safety for many years, and the present safety organization has been in effect for the past 7 years. It was formed on the theory that safety depends largely on the fundamentals of education, supervision and discipline, and that it is the duty of the Safety Department and executives, as wholesalers, to sell safety in all its various forms to the foremen, fire bosses and others in charge of men, so that they, as distributors, may get it into the minds of the consumers (the workmen).

The organization is headed by the Supervisor of Safety and Compensation, who reports directly to the General Manager. His department consists of Safety Inspectors at all collieries, the Chief Surgeon and his staff of doctors.* At the time the organization was formed, several years ago, and after a study of physical conditions and practices, a meeting of all general foremen was called to meet with the Safety Department, and in a conference lasting several days adopted a standard of "Safety Hazards and Guide in Making Safety Inspections." All items were passed unanimously before being included in the "standard," thereby giving all foremen a definite part in the building of the guide governing the inspection of the property directly under their supervision. The safety standard is prefaced by the general instructions: "These rules are intended to supplement, not supplant in any manner or form any of the requirements or provisions of the mining laws of the State of Pennsylvania." This standard is revised annually and is placed in the hands of all foremen.

Great care was taken in choosing the Safety Inspectors, who are equal to the foremen in mining experience and are well trained in safety. In addition to investigating all accidents, the Safety Inspector makes

* The Chief Surgeon and the doctors are salaried employees but also have private practices.

daily inspections underground. With no previous notice, he appears at the mine as the shift is about to start and selects a fire boss or section foreman to accompany on his daily inspection. The fire boss covers his regular district accompanied by the Safety Inspector, who not only notes any unsafe or substandard practices or conditions but observes the attitude and action of the fire boss toward safety as well. He discusses

Report Covering Safety Inspection of Workings

under the supervision of

Mine Foreman __________ Asst. Mine Foreman __________ Fire Boss __________ Pennsylvania Colliery, Nos. 5 and 10 Slopes, No. 2 Level, Nos. 177 and 185 Tunnels, Nos. 4 and 8 Veins South Dip, No. 9 Vein North and South Dips.

Report No. 555 *June* 28, 1938

CRITICISMS	*COMMENTS BY MINE FOREMAN*
1. *No.* 10 *Slope, No.* 182 *Tunnel, West* 8 *Vein Gangway, Breast No.* 4 *and* 5, *Pitch* 50 *degrees. Would recommend that steps be placed, not more than* 3 *ft. apart, from the monkey heading to breast manway.*	1. *Steps are now placed at proper distance in these breasts. Fire Bosses have again been instructed to see that all steps are properly placed to prevent persons slipping or falling.*
2. *No.* 10 *Slope, No.* 182 *Tunnel, Rock Hole No.* 17. *Air line to monkey heading not insulated.*	2. *The Fire Boss and Machinist Boss were severely reprimanded for this offense.*
3. *No.* 12 *Vein Surface Counter, No.* 1 *Lift, Breast No.* 1. *Top coal and top rock over edge of platform should have been propped or taken down before I made examination.*	3. *Place was immediately propped. This is first offense for this miner, who was reprimanded and warned that suspension would follow a repetition.*
4. *No.* 5 *Slope, No.* 5 *Plane, No.* 1 *Level East, Pillar Skip No.* 7½. *Miner tested top very poorly—Insufficient area.*	4. *This miner was given a severe reprimand and again shown the proper method of testing top.*
Signed	Signed
__________ Safety Inspector	__________ Mine Foreman

Copies to:—
Supervisor of Safety
Superintendent
Mine Foreman
Safety Inspector

FIG. 2.—SAFETY INSPECTOR'S REPORT.

conditions and practices with the fire boss, drawing him out as well as giving opinions and advice. He observes the miners at work and stops a few minutes with each one to discuss the conditions of his working place, proper methods of testing, proper methods of storing, preparing and firing of explosives, etc., and endeavors to build up a greater safety consciousness and cooperation with all in the section.

At the completion of his inspection, the Safety Inspector notes any criticisms or recommendations he may have on the left side of a report sheet (Fig. 2). He discusses these with the mine foreman, who, on the right side of the report, indicates the disposition of the criticism or recommendation. The report is then typed and copies are sent to the Superintendent, Supervisor of Safety and mine foreman. This report is carefully examined by the Supervisor of Safety, and any criticisms or recommendations of the Safety Inspector improperly or unsatisfactorily answered by the mine foreman are immediately taken up with the Superintendent. The Supervisor of Safety and Compensation also visits all collieries and makes joint inspections with the Safety Inspector and fire boss.

Every foreman and fire boss has a card on file in the offices of the Safety Department, on one side of which are recorded the result of the safety inspection, date and number of criticisms, and a summary of these, and on the other a record of all accidents. These cards are valuable in comparing district with district as well as period with period in the particular district.

First-aid teams are maintained at each colliery and all first-aid training is by a corps of company surgeons. The U. S. Bureau of Mines Manual of First Aid Instruction is used as a text and from this text a series of 12 lessons has been made up and approved by the Chief Surgeon. By this method all surgeons teaching first aid are using a standard text and the schedule is such that all first-aid classes are being taught like subjects at the same time. First-aid contests are held from time to time. Emergency hospitals are maintained on the surface and on each level underground at each colliery, wherein any injured workman may receive prompt and efficient attention.

Regular teams, certified in mine rescue by the U. S. Bureau of Mines, are maintained at each colliery and others are at all times in training under the direction of the Supervisor of Safety. The Susquehanna Collieries Co. maintains four well equipped mine-rescue stations with a certified trained man in charge of each. The latter makes a monthly report (Fig. 3) of training, condition of apparatus, amount and location of supplies and certifies that every machine in his charge is ready for instant use. The colliery Safety Inspector also inspects the rescue stations and equipment and makes a monthly report to the Supervisor of Safety. It is estimated that in an emergency the total equipment and supplies of all rescue stations can be assembled at any location on the property within a period of two hours.

Under the direction of the Superintendent, monthly safety meetings are conducted at all collieries. These are attended by all foremen, mechanical and electrical employees, and at each meeting two or three selected workmen. All accidents at the colliery are analyzed, every

effort being made to fix the cause, and steps are then taken to prevent similar accidents in the future. Records of these meetings are kept and copies are sent to the Superintendent and Supervisor of Safety and Compensation. The Safety Department prepares a statement, with blueprints of location and detail, of all serious and fatal accidents for the Safety Inspectors and Superintendents. These are carefully examined and discussed at the colliery safety meetings.

Fire bosses hold daily 5-minute safety meetings with their men at fire-boss stations in the mines. One day the men will be warned of bad

SUSQUEHANNA COLLIERIES COMPANY

Monthly Report—Mine Rescue Apparatus

Colliery: ______________ Date of Inspection: ____________________

Mach. No.	Type	Location	Condition as Found:	Repairs Made:	Conditions as Left:

Location and Condition of Gas Masks.

Supplies & Repair Parts on Hand:

Supplies & Repair Parts Needed:

ADDITIONAL RESCUE EQUIPMENT

Inspected by:

Mine Rescue Instructor

Colliery Safety Inspector

FIG. 3.—REPORT ON MINE-RESCUE APPARATUS.

top conditions and the importance of careful and frequent tests; another day the fire boss may discuss the proper use of explosives.

Bulletin boards are maintained at all collieries with many *Caution*, *Warning* and *Danger* signs within the mine. In each section of the mine is maintained a slate showing the number of working days of that section

since the last lost-time accident. Every effort is made to keep both the foremen and workmen safety conscious; one of the major plans in use for the past several years is as follows:

Each colliery is divided into units. A unit may be an opening, openings or subdivision of an opening. Each unit is further divided into

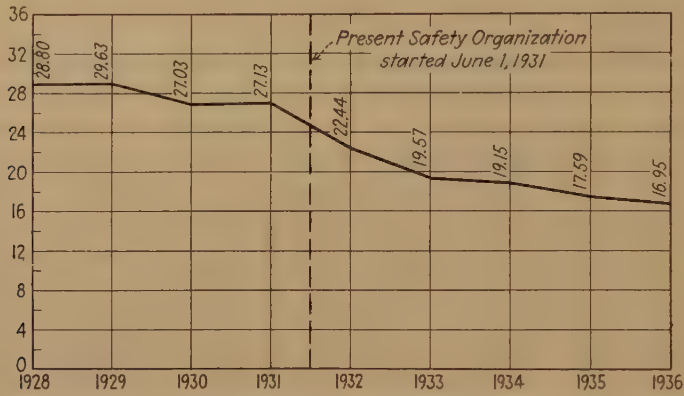

FIG. 4.—COMPENSABLE ACCIDENTS PER 100,000 NET TONS FRESH MINED, SUSQUEHANNA COLLIERIES CO., 1928–1936.

fire-boss or other comparable sections, and the sections within each unit compete for the best safety record of the unit covering a three-month period. At the close of the three-month period, the best safety section of each unit is determined and each employee of this best safety section

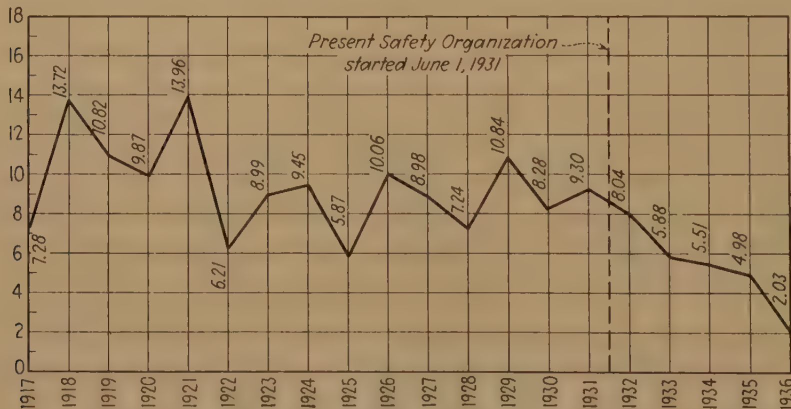

FIG. 5.—FATAL ACCIDENTS PER 1,000,000 NET TONS FRESH MINED, SUSQUEHANNA COLLIERIES CO., 1917–1936.

who did not suffer a lost-time accident during the period is eligible to draw for a substantial cash prize. Foremen and office employees are not included in the contest, but as members of the Safety Committee attend safety banquets and picnics from time to time.

Many of the other companies of the anthracite field provide similar or other means of keeping safety the first consideration in the minds of their foremen and workmen. One of the larger companies issues a monthly safety paper for distribution, and annually sends the colliery superintendent, the mine foreman and two or three sectional foremen with the best safety record to attend the meeting of the National Safety Congress.

Colliery safety flags and the distribution of buttons to those having good safety records are used with good results.

Many collieries have availed themselves of the U. S. Bureau of Mines cooperative training and have their employees 100 per cent trained in first aid.

Three chapters of the Holmes Safety Association are maintained within the anthracite field and report most excellent attendance.

A survey of the larger anthracite companies indicates that the safety organization is a combination of the safety and operating departments, that complete harmony and cooperation exist between them and that the results substantiate the belief that much progress has and is being made toward greater safety. The two curves (Figs. 4 and 5) showing accidents per ton mined by the Susquehanna Collieries Co. indicates the considerable progress made by that company.

DISCUSSION

(*Gordon MacVean presiding*)

R. D. Currie,* Wilkes Barre, Pa.—In anthracite, as in all other types of underground mining, roof-fall accidents account for a large majority of the injuries, and it is only natural that this phase of accident-prevention work should be given serious consideration.

A number of years ago the Pennsylvania Department of Mines organized a campaign to reduce this type of accident, primarily by giving it more publicity. A bulletin is issued each month showing the standing of the mining companies based on "direct" roof-fall accidents and tonnage. This effort apparently is getting good results at some of the mines, although the records show that a small group of companies, perhaps 8 or 10, generally hold one of the first 8 or 10 positions in that report.

A few companies and a few officials have been able to show outstanding records in the prevention of roof-fall accidents.

Mr. Brehm has pointed out that anthracite mining conditions physically are considerably different from the usual bituminous conditions, although a geologic column of some of the mining districts of West Virginia would make us feel that 13 to 16 coal beds is a mere trifle, and cross sections of some of the bituminous mines in the state of Washington would remind us that "soft" coal is also able to stand on its end—"if not on its own bottom."

In general, the hazards of anthracite mining, as depicted by accident rates, follow closely the same lines as the hazards in bituminous mining, with three exceptions, explosives and blasting, handling of material, and haulage.

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* District Engineer, U. S. Bureau of Mines.

Blasting is a major operation in anthracite mining and much larger quantities of explosive are used in these mines than in bituminous mines. It is not uncommon to use a pound of explosives per ton of coal; shooting off the solid is customary; multiple shots, frequently 10 to 15, are fired, and the miners generally fire their own shots at any time during the shift.

At the present time a proposed standard in the form of "Recommended Practices" is being prepared for the use of explosives in anthracite mines, and it is hoped that an organized effort on the part of all companies will be made to adopt and follow out these standards.

Accidents from handling material, especially from handling timber on the pitch, account for many serious injuries, but very little has been done in the way of organized or even individual effort to remove this cause of accidents. The Pennsylvania mining law requires that timber and supplies be furnished to the mines "as near to their working place as they can be transported in ordinary mine cars," but Fig. 1 shows that this is frequently several hundred feet vertically from the point where it must be used.

Virtually every stick of timber, lagging and plank used in these steep breasts is lugged or tugged up the pitch by brute strength with resultant hernias, strained backs, and frequently more serious results. An organized effort to establish a method whereby this hazardous and strenuous work can be done mechanically is inevitable.

The accident rates from haulage in anthracite mines, as compared with bituminous mines, are low, and this can be accounted for in two ways. The ton-miles in these mines is much lower than in bituminous mines, and in the Southern, and Middle field there is no tract or haulage in the working places. In the Northern field the ton-miles would probably compare favorably with the bituminous mine. The real reason for the low accident rates in haulage in these mines, however, is the strict enforcement of rules and regulations. It is the exception to see anyone other than the crew riding on a locomotive or trip.

Mr. Brehm has referred to the Cooperative first-aid training being conducted by the Bureau of Mines. In 1933 we brought a mine rescue car into the region for the first time since 1914—and since January 1933 that car has been in almost every mining community in the region and there have been approximately 35,000 first-aid certificates issued to anthracite miners who have completed the course, and 700 instructor's certificates to those who have qualified for them.

This safety education is being offered to every miner and mining company in the Anthracite Region, and is perhaps the largest, if not the most important, organized safety work being done in the region today.

A number of companies have taken advantage of the accident-prevention course of the Bureau with excellent results and it is hoped that an organized effort in safety education of mine officials will soon result in a more rapid reduction of anthracite-mine accident rates.

Mr. Brehm represents that small group of safety engineers on whose shoulders rests the responsibility for this tremendous task, but with organized effort it will be done.

W. G. Metzger,* Scranton, Pa.—The holding of a conference of all general foremen and the members of the safety department to adopt rules governing the safe conduct of the workmen appeals to me as an excellent idea; the foremen, having assisted in setting up the safety rules, will be much more interested in their enforcement. The annual revision of the rules keeps them up to date.

The selection of safety inspectors whose experience in mining is equal to that of the mine foremen is a wise precaution. They must criticize unsafe conditions and unless

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* Hudson Coal Co.

the mine foremen know they have experience equal to or greater than their own the criticism will have little effect.

The safety inspector making an inspection without previous notice tends to prevent the foreman from preparing his section for the inspection. I have, however, known of a colliery foreman sending a man ahead of the inspector to warn the others of a safety inspection. This enables the men to clean up any minor violations of good safety practice but will not afford sufficient time to remedy major unsafe conditions.

The safety inspector's report is standard with a few of the anthracite companies and covers details necessary to make the inspection of value and effective. It provides a written record of the interest of the foreman in the safety of his men. However, I have found that some sectional foremen keep sections in perfect safety condition and without criticism for unsafe conditions and yet their accident record does not reflect this interest in safety, probably because of their inability to teach their men to do their work safely, not having the ability to instill in their men their own enthusiasm for safety consciousness. Some foremen hesitate to deprive a man of his earning power by strict, fair administration of discipline. They do not fully realize that their failure to discipline at the proper time may deprive the man's family of his earning power through a fatal accident.

A record of the safety performance of a foreman as well as operating performance is a great help when it is necessary to select a new foreman. First-aid training is one of the basic principles in teaching the men to do their work safely. I have never heard of a company in which the safety record did not improve materially after the men had been trained thoroughly in first aid.

The maintenance of trained mine-rescue teams is good insurance because it affords prompt acting in fighting fires as well as rescue of those exposed after an explosion.

Analysis of accidents at safety meetings of officials keeps interest in safety very keen. The fire bosses' talks on different unsafe conditions relay safe practice to the men.

Although bulletin boards do not attract the attention of the men to as great an extent as they should, they are an important part of safety working giving to the workmen every opportunity to be safety minded whether they take advantage of it or not.

Inspection of mine-rescue apparatus and gas masks is necessary, because unless this equipment is kept in condition it creates a false security. If it were not usable in an emergency it would be infinitely worse than no equipment at all because in this case help would be asked of companies having the rescue equipment. Then too it would expose the men on the rescue team to great danger, as their lives depend on the equipment being in good usable condition.

The competitive awards to different collieries, veins and sections of mines keep alive the interest in the safety movement.

Ground Movement and Subsidence Studies in Mining Coal, Ores and Nonmetallic Minerals

A Review of the Work of Fifteen Years and Suggestions for Future Studies

By George S. Rice,* Member A.I.M.E.

(New York Meeting, February 1937)

The A.I.M.E. Ground Movement and Subsidence Committee, proposed in 1920, held its first technical meeting in February 1923, under the able chairmanship of Mr. H. G. Moulton. The following list of papers and discussions, indicating the scope of the problems and relation to various conditions and method of mining, were given at that meeting by the distinguished mining engineers: J. Parke Channing, on Subsidence at Miami, Arizona; Howard N. Eavenson, on Mining an Upper Bituminous Seam after a Lower Seam Has Been Extracted; J. J. Rutledge, on Subsidence in Two Oklahoma Coal Mines; Benjamin F. Tillson, on Caving Cracks at the Franklin Mine, New Jersey; Dr. F. W. McNair, on the Dome Theory; William Kelly, on disadvantageous effects in Mining and Back-filling Working Upward a Steep-Fitching Iron-ore Bed; Eli T. Conner, on the good results of Back-Filling by Hydraulic Means in Preventing Destructive Subsidence in the Anthracite District; Thomas H. Claggett, on the merit of Systematic Pillar Drawing in Preventing Cracks in an Upper Seam; Henry H. Otto, successful results in the anthracite district in Mining Lower Seams before Upper Seams; Douglas Bunting, on Leading Factors Incident to Successful Mining of an Overlying Seam; Edwin Ludlow, on there being "less damage to buildings when there was complete removal of coal with complete packing than when room and pillar is used without extracting the pillars"; Edward O'Toole on "bumps" in eastern Kentucky mines, which he thought were the result of geologic stresses: H. A. Buehler, on Upheaval of Surface in an Illinois Case in Advance of the Face; and R. Dawson Hall, on The Mechanics of Subsidence from Mining. The author of this paper discussed various kinds of problems of ground movement and subsidence in both metal mining and coal mining; the production of great landslides by workings near an outcrop, as at Turtle Mountain, Alberta, Canada; and general

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theories of ground movement. Tentative forms were presented for the systematic gathering of data on ground movement and subsidence.

The foregoing subjects, concerning the mining of coal, iron, copper and zinc, indicate how important was, and still is, the problem of ground movement in relation to methods of mining not only of the foregoing named minerals but any mined, as discussed in later papers, including those extracted by wells such as petroleum, sulphur and salts; the effect in causing loss of the mineral mined (and overlying it); the damage from subsidence to surface improvements of all kinds and the legal questions involved.

The extensive list of problems considered in that first meeting of the committee shows the importance of the general subject from a technical standpoint to mining engineers and to owners of mining properties because of legal liability and operating cost. Before the appointment of this committee there had been some very important cases in which members of the Institute had been concerned but which had not appeared in its TRANSACTIONS. For example, the many instances of subsidence in the anthracite district of Pennsylvania, causing much surface destruction, culminating in the case of subsidence at Scranton described in the classic report of Conner and Griffith, which was published by the Bureau of Mines (*Bulletin* 25 in 1912).

In 1911 the State of Pennsylvania appointed an Anthracite Mine Cave Commission with which the U. S. Bureau of Mines cooperated in making large-scale tests on the bearing strength of full-size timbers and cribs and also backfilling supports and their compression under load in the 10,000,000-lb. capacity machine of the Bureau of Standards. The report thereon rendered to the Legislature in 1913 was not published by that body, but later was abstracted in Bureau of Mines *Bulletin* 303, with review by G. S. Rice on compression tests (1929).

In 1915 an important study of subsidence from coal mining was undertaken in Illinois under a cooperative agreement between the Engineering Experiment Station of the University of Illinois, the Illinois Geological Survey and the United States Bureau of Mines. The first publications under this cooperative status were Surface Subsidence in Illinois Resulting from Coal Mining (Illinois Geological Survey *Bull.* 17, 1916) by L. E. Young, and *Bulletin* 91 (1916) of the University of Illinois, by L. E. Young and H. H. Stock. The latter included a comprehensive review of important literature on ground movement and subsidence from coal mining, which has been published in Europe and America, and reported observations made by the writers in Illinois and presented theories of the mechanics of ground movement and subsidence.

This was followed by another cooperative investigation carried on by engineers of the U. S. Bureau of Mines in the Illinois coal field, beginning in 1916. Surface monuments were established in three typical districts

in Illinois, (1) in which the method of mining was ordinary room and pillar; (2) panel room and pillar; and (3) longwall in northern Illinois. The latter was of particular interest because a place was selected in which two longwall mines were approaching one another under a large school building; the "draw" in advance of the face was determined and time-subsidence records were made. The foregoing data appear in a report on Subsidence Due to Coal Mining in Illinois, by C. A. Herbert and J. J. Rutledge (Bur. Mines *Bull.* 238, 1927, with foreword by G. S. Rice).

Data Gathered by Subcommittees

Following the discussions at the meeting of the Committee in 1923, it was decided that it was important to gather all precise data possible to make these matters of record for future analyses. To further this project three subcommittees were appointed by Chairman H. G. Moulton, "to collect data of every conceivable nature which might serve as a basis for propounding new theories, for locating surface works or for appraising damage to adjacent properties": (1) underground metal-mining practice (2) open-cut practice and (3) bituminous coal mining and (4) anthracite.

At the annual meeting in February 1925, Mr. Louis C. Cates, who was chairman of the subcommittee on open-cut practices, gave a progress report, "Factors Affecting Bank Slopes in Steam-shovel Operations" (*Trans.* A.I.M.E. (1926) **74**, 818–825).

Subcommittee 3, on bituminous mining operations, under the leadership of Mr. Eavenson, prepared a comprehensive report[1] with a bibliography, which was presented at the meeting in February 1936.

In this report evidence of subsidence was classed under four heads:

1. Those due to squeezes or subsidence when pillars have not been withdrawn.
2. Those in upper seams due to mining seams beneath them.
3. Those due to mining by the room-and-pillar system (pillars being withdrawn more or less completely).
4. Those due to mining by the longwall system.

One of the conclusions drawn by subcommittee 3 in reference to class 3 from the evidence presented was that surface subsidence from room-and-pillar work did not extend beyond the excavated area. This conclusion was much questioned in the subsequent discussion. Although it appears to have been supported by the studies of Herbert and Rutledge, previously referred to, in certain Illinois mines that used the panel system with surrounding pillars, it has not been found in later studies to be true where all pillars are withdrawn in a long break line, making the effect like that of longwall retreating. An example of this was found by Newell and Plein at Merrittstown, Pa., as shown by the measurements reported

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[1] Report of Subcommittee on (bituminous) Coal Mining (1926) **74**, 734–796.

in their paper on Subsidence at Merrittstown Air Shaft near Brownsville, Pennsylvania, in 1934 (*Trans.* **119,** 58–94).

The other subcommittees did not render reports, but from time to time papers covering various phases of ground movement and subsidence from the mining or extraction of various minerals from the ground have been submitted to the Institute and appear in its TRANSACTIONS.

Manifestly, it is impossible to consider specifically each of these papers and discussions in this review or even to give their titles, but it seems useful to tabulate by groups the 65 papers and discussion that have been presented in the 15 years the committee has carried on (see accompanying table).

Classification of A.I.M.E. Papers, Discussion and Reviews on Ground Movement and Subsidence, 1923 *to* 1937, *Inclusive*[a]

Underground mining:	
Coal mining, subsidence and normal movements	13
Coal mining, instantaneous violent outbursts of gas (due primarily to natural conditions but lessened in dangerous effect by suitable mining methods and shock blasting)	4
Coal mining, bumps of violent character	5
Iron mining, subsidence and movements	5
Copper and other metal mining, subsidence, movements and rock bursts in deep mines	9
Potash mining, convergence studies	2
Well extraction, subsidence of surface:	
Petroleum	2
Salt	1
General to different kinds of mining:	
Convergence of roof and floor measurements	6
Laboratory testing of strength of specimens and of engineering models	7
Legal questions of damage from subsidence and laws	3
Reviews, annual and general	8
	65

[a] There are a number of duplications in this total as many of the papers and discussions concerned several subjects.

It is natural that the largest number of papers should deal with coal mining, for two reasons: (1) it is the most extensive specific mining industry and (2) most of the mines are in densely settled parts of the country and there are many buildings, cultivated farms and towns to be affected. Metal mines are usually in mountainous regions and less often are valuable structures and surface improvements affected. For all kinds of mining, however, more data are needed for conclusions, including subsidence data on extraction of sulphur, petroleum and salt by wells.

Studies related to mechanics of ground movement, convergence data in mines, and laboratory testing of the bearing strength of specimens and models have been increasing in number during recent years.

The Writer's Views on Information in These Papers and Discussions

1. Ground movement and subsidence without reference to magnitude or severity are independent of the mineral mined; it is primarily a question of underground excavation.

2. The geologic formation, including its dip, the strength of the enclosing rocks, the size of the excavation, its depth below the surface and the presence of intersecting major faults are the more vital factors.

3. The method of mining and the amount of back-filling are factors in controlling ground movement and subsidence effects.

4. More or less subsidence of surface inevitably follows complete extraction, but the amplitude of subsidence varies with the method of mining, the extent of back-filling, if any, the character and depth of overburden, and the time that has elapsed since the mining was completed. The time might be a matter of months in subsidence from shallow mining, or of many years in deep mining where partial support is given by pillars, or by rocks from caving roof.

5. The relation of the amplitude of subsidence with reference to the thickness of excavation varies with the depth of the excavation. In general the amplitude decreases with depth and also with increase in the angle of dip from the horizontal. This relationship is variable, depending upon the amount of the filling material and whether the underlying rocks are rigid, like sandstone, or semiplastic like shales, which readily compress and fold, but there is no such thing as a "harmless depth," as believed a century ago in Europe; however, with great depths, the wider surface area that may be affected tends to reduce the percentages of amplitude of subsidence, to thickness of excavation, to a very small figure, as shown by studies made by the late Prof. Henry Briggs, of British and German coal-mine subsidence data, which he compiled and presented in his classic book on Mining Subsidence (pp. 94–106).

Conditions are so different in different coal fields of the United States and mining methods vary so greatly, that formulas, while applicable under identical conditions, do not have general application.

Metal-mine Subsidence

In formations like that of the iron ore at Birmingham, Ala., which lies like that of a dipping coal bed, the subsidence effects should be similar for the same percentage of extraction per acre.

In the steep-dipping iron-ore beds in the Lake Superior region, although the iron ore generally lies between well-defined walls, the ore bodies usually occur in thick lenses. The subsidences from mining in the district often are very great, in spite of a certain amount of back-filling with waste from the surface. The subsidences are sometimes greatly affected by the presence of dikes and faults. One of the most

extraordinary cases was reported in the paper by C. W. Allen[2] on a subsidence in the Athens mine at Negaunee, where there had been an excavation in a lens of ore 250 ft. wide, 600 ft. long, and 200 to 250 ft. deep; the bottom of the excavation was about 2200 ft. deep below the surface. The area that caved was bounded by a vertical diorite dike and, 350 ft. distant, by a so-called "fault dike." The block of jasper capping 1900 ft. deep unexpectedly caved to the surface on June 19, 1932.

Ground Movement in Iron Mine Where Surface Subsidence Has Not Yet Occurred

In contrast with the subsidence at Negaunee, there is the ground movement at the Brier Hill iron mine, Michigan, reported by the writer.[3] Beginning in 1913, there was danger that ground movement might cause subsidence under the tracks of the Chicago and North Western Railroad, and a commission was then appointed for observation. In 1916 a majority of the Commission reported there was a hazard of subsidence if the mining continued. However, the writer suggested there was no immediate danger and proposed that drill holes from the surface be put down and records kept of the successive cavings and thus warning given before a dangerous condition occurred.

From the iron-ore lens under and adjacent to the tracks up to 1921 there had been mined 871,000 long tons of iron ore having a volume of about 7,200,000 cu. ft. The top of the ore body lay at a depth of less than 600 ft. from the surface. As determined by the test boreholes, the caving by the end of 1921, when the mining in the vicinity was practically finished, extended to the top of the rock formation, about 70 ft. from the surface. However, to the present time (1938) caving has not broken through to the surface.

At this place the rocks of jasper falling from the hanging wall and more or less filling the cavity were so strong that they probably bridged over the main cavity. Waste rock material from the surface had been used to more or less fill the stopes and, in all probability, overlying drift, clay, sands and gravel had tended to fill cracks and crevices. However, this cannot be certain, and there may yet be a surface subsidence.

Subsidence from Copper Mining

Space will not permit consideration of the many cases of metal-mine subsidence presented in the various papers and discussions, except in unusual instances of the porphyry mass copper mines at Globe, Ariz., where the block caving method of mining was developed with most

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[2] C. W. Allen: Subsidence Resulting from the Athens System of Mining at Negaunee, Mich. *Trans.* A.I.M.E. (1934) **109**, 195.

[3] G. S. Rice: Ground Movement from Mining in Brier Hill Mine. *Trans.* A.I.M.E. (1934) **109**, 118.

favorable results in securing maximum extraction, as described in a paper by J. Parke Channing (cited on page 140) and one by the writer in 1923 (TRANSACTIONS, vol. 69). Under the condition no attempt could be made to prevent surface subsidence, nor was it necessary, because the value of the surface structures other than a shaft house was small. The chief point of interest from a ground-movement standpoint was the flat angle, 42°, at which the material slid into the caving area, undoubtedly because of a fault plane.

Ground Movements Underground

Surface subsidence results from underground movements, of course, and such movements are universal in mining, beginning when excavation begins, with more or less squeezing or deformation of the pillars, if the pillar system is used, and of the face of the adjacent solid mineral being mined; also movements causing fracturing, bending in the roof or hanging-wall formation and stresses in the floor or footwall. As these movements continue the roof or hanging wall will either break or bend and the floor or footwall upheave.

For normal conditions with certain methods of mining these movements assist in breaking the coal or ore, as in longwall mining and the pillar-retreat system, by throwing weight on the working face, and similarly in the caving systems in metal mining. On the other hand, the stresses and strains in the roof or hanging-wall formations in bending or breaking present the greatest problems in mining in designing supports suitable to the conditions in the working places for efficient and safe mining.

Research on Roof Supports

Until the last decade, the design of roof supports has largely depended on the method of "trial and error," but in recent years much research has been done in England and Germany and to a certain extent in the United States, as indicated by the papers and discussions presented to the Institute during the past several years; this research has been intensified by the rapid increase of mechanization in the working places, especially in coal mines of the United States, by the introduction of large cutting and loading machines requiring much operating space at the working face. These machines have decreased the accident rate per ton but tend to increase the individual risk, which has led to much research to provide efficient roof support.

The researches in coal mining, initially made in England to determine the effects of movement of roof and floor and compression of pillar or the adjacent solid coal, were directed to convergence studies; to determining how the roof fractures adjacent to and over the solid, the angle of draw, the breaking planes in the roof stratum, the effect of different supports,

timber and pack walls, and the unit loads carried by the supports. The writer, in a visit made to certain English mines in 1928, was impressed with the convergence testing carried on by Dr. A. Winstanley, of the Safety in Mines Research Board. The Bureau of Mines, which had already been studying the compressibility of coal in place in the Experimental Mine, took up the convergence studies and later applied the method in retreat work in a highly mechanized mine in the Pittsburgh district. Subsequently the writer recommended such testing for a mine subject to severe bumps in Harlan County, Kentucky, and for a potash mine at Carlsbad, New Mexico. Meantime, Professor Landsberg took up a similar study in a central Pennsylvania field. These studies have been described in papers given in 1937 and 1938.

In addition to measuring convergence of roof and floor in England and Germany, hydraulic pressure-recording props were developed to determine the local unit roof pressures in working places. It is hoped that similar attempts to obtain pressure data will be made in mines in this country to supplement convergence studies. Necessarily they involve much expense if properly done, as groups of recording props are required, rather then individual ones, as hitherto tried.

Mechanics of Roof Strata in Breaking

For generations the manner in which roof behaves over an extensive mine excavation was studied and many theories were presented. The early studies were concisely reviewed in 1916 by Dr. L. E. Young, of our committee, and Professor Stoek, in their classic study on Subsidence from Mining,[4] and later by Prof. Henry Briggs in his book on Mining Subsidence, published in 1929.

The writer, then acting as a representative of the U. S. Bureau of Mines in an advisory capacity to the Illinois studies, and later in his paper given before this committee in 1923, presented certain theories of the difference in roof behavior in longwall and room-and-pillar working. Under the first method, the roof should be made to bend or break by tension above the face, if possible; under the second, the roof should be considered as a beam or plate supported chiefly by the pillars until retreat is begun.

Angle of Draw

In connection with subsidence studies, data on the "angle of draw" were among the early objectives of investigators. As first employed it meant the pulling by tension or the sliding of strata from above the unmined coal or other mineral toward the mine excavations. This has been discussed in various committee papers as well as in many early publications.

[4] Univ. of Illinois Eng. Expt. Station, *Bull.* 91.

It has been shown beyond doubt that so-called "positive draw" always takes place with the system of advancing longwall, and also with the system of retreating faces that have long break lines. The angle of the line of draw, or, more properly, the plane of draw, from a vertical above the edge of the excavation, varies with the structure of the rocks, the height of excavation and the rapidity of its advance.

So-called "negative draw" occurs in panel work in coal mines and sometimes in other kinds of mining where shear, perhaps due to rock joints or fault planes, causes the rocks to shear at the edge of the excavation.

The line or plane of draw is only theoretically straight from the edge of the excavation to the furthest point or edge of subsidence. Practically, it is irregularly curved, usually extending from over the working face, where the individual roof rocks break off, then back over the solid in irregular curves, varying with the formation. Usually, as it approaches the surface, where there is a layer of clay or sand, the line or plane approaches verticality.

Mechanism of Bumps in Coal Mines

The study of "bumps" in some coal mines that have special conditions have thrown light on the general problem of mechanism of roof movement by comparison with mines that do not have these special conditions and do not experience violent bumps.

These special conditions are summarized in the writer's paper[5] written in 1935, which gives references to previous committee papers, including that by Walter Herd in 1929, and T. L. McCall in 1934 on bumps in the Springhill mine, Nova Scotia, and by J. F. Bryson in 1936 on prevention of bumps in a Harlan County (Ky.) mine.

Briefly, conditions where bumps occur are principally the presence of a strong, thick, massive stratum, generally sandstone, immediately over or close above the coal bed; that is, without a thick cushioning layer of shale between, and may occur where the pillar system of mining is used. The danger of violent bumps increases with the depth or weight of the overburden. The breaking of this strong stratum as the span of an excavation increases in drawing pillars causes violent bumps from time to time, crushing and bursting pillars, not always the one bordering the goaf. If complete extraction of coal is to be attained, the remedy is longwall with pack walls or else rock-filled cribs to control the subsidence of the roof.

Theory of Cause of Bumps

The writer has put forward the theory that it is the breaking of this strong stratum spanning a wide excavation, by compression at some plane considerably above the bottom of the stratum, that causes a

[5] G. S. Rice: Bumps in Coal Mines. *Trans.* A.I.M.E. (1936) **119**, 11.

"shock wave" to strike one or more pillars, the wave traveling in a curving path deflected downward by the solid buttress of strata beyond. He has postulated a theory of preventive means on the fact that: (1) rocks like sandstone have relatively low tensile strength—in fact, usually have joint planes more or less vertically crossing the stratum; (2) as a span increases, the stratum bends more or less downward, but generally not far enough to rest on the floor of the goaf. Thus the upper part of the stratum, like a beam spanning the goaf, is thrown into compression, and when the compressive strength of the rock is reached, which in rocks like sandstone or conglomerate is very great,[6] a sudden crushing occurs. This rupture may cause a strong shock wave, referred to above, which on reaching pillars bordering or near the goaf causes one or more to burst violently.

Proposed Means of Preventing Bumps

The writer's proposal for preventing the occurrence of bumps or minimizing their effects was described in the paper given before the committee in 1935 (see footnote 5). Briefly, it is to try to make the strong stratum, always present where bumps occur, generally a massive sandstone: (1) acting as a cantilever extending over the solid coal, to bend or else break by tension, above the longwall face, or above the retreat pillar line, assuming in the latter case that the pillars are left large on the advance work and contain at least 80 per cent of the coal area; (2) to accomplish the bending or breaking in tension, instead of compression, above the edge of the solid by cushioned supports in the goaf, rock-filled cribs as previously mentioned.

Trial of this method, which had successful results, was made in the Mary Helen No. 3 mine, Harlan County, Kentucky, as described by J. F. Bryson.[7]

Rock Bursts in Metal Mines

"Rock bursts" are similar in violent effects to bumps, but in the opinion of the writer are usually different in that for the most part they are caused by the overloading of a strong pillar, which bursts under the pressure, causing accidents: "bursts" usually do not occur until the depth from the surface exceeds 2000 or 3000 ft. or pillar pressures exceed 4000 or 6000 lb. per square inch.

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[6] Tests by the Bureau of Mines on specimens of sandstone from the Pittsburgh upper Carboniferous strata have shown a compressive strength of 15,000 to 20,000 lb. per sq. in. On the other hand, the tensile strength is relatively small. Troutwine's Engineer's Pocketbook gives a range of only 105 to 590 lb. per square inch.

[7] J. F. Bryson: Method of Eliminating Coal Bumps or Minimizing Their Effects. *Trans.* A.I.M.E. (1936) **119**, 40.

Further Developments in Preventing Bumps in Harlan County Coal Mines. A.I.M.E. *Contrib.* 107 (1937).

In the past, these have occurred in the deep, steep-dipping copper mines of Lake Superior, in which pillars were used between narrow stopes or chambers. In recent years they have been largely prevented by extraction on a retreat system.

In the gold mines of South Africa and Kolar, India, however, rock bursts have been much more frequent and severe and increasingly so with great depths. Judging from discussions in technical journals, apparently there is a difference of opinion as to whether back-filling, with sands run or lowered into the mines from the surface, was beneficial or had little value. Those of the latter opinion think that the remedy is in completely taking out all pillars and making the roof break and fill the stope behind the extraction face.

The extremely difficult conditions found in South Africa and methods of meeting them are described in an illustrated report made by the Association of Mine Managers of the Transvaal (Some Aspects of Deep Level Mining on the Witwatersrand Gold Mines with Especial Reference to Rock Bursts, 1933).

Seismograph stations have been established on the surface both on the Rand and at Kolar, apparently with but little success in giving warning of specific occurrences, inasmuch as the general movements of ground in extensive mines cause a confusion of records. So-called "sag-meters" set up in stopes, to indicate compression at the face of a pillar are said by some South African engineers to be effective in giving warning of threatening rock bursts, but have not received general acceptance. They are a form of convergence recorder, and some have warning devices attached. The accident record in South Africa has shown marked improvement, but accidents still occur from rock bursts.

Convergence Studies in a Potash Mine

The United States Potash Co. is concerned in developing the best method of mining that will permit the largest recovery of potash from its mine at Carlsbad, N. M., on a Government lease. In this effort the Geological Survey and the Bureau of Mines are cooperating. The potash company has been considering the adoption of a plan somewhat similar to that employed in preventing bumps, except that instead of using cribs it may utilize (the writer has suggested) the highly plastic properties of potash salt in small pillars, which would deform but not break. Rock-filled cribs and rocks for filling them would be expensive at Carlsbad.

As a preliminary to this the Bureau has carried on tests on the bearing strength or plasticity of potash salt and of the rock-salt roof by specimens tested at Pittsburgh under the supervision of H. P. Greenwald, as described in a paper given in 1937 before this committee. As a further

step, C. A. Pierce has been carrying on convergence studies in the United States Potash Company's mine and has given a progress report.[8]

Tests of the Strength of Materials and of Models

Laboratory Tests.—The laboratory testing conducted at the Pittsburgh Experiment Station on the strength of specimens, coal, roof material, and potash salt has been already touched upon. It has been shown with great definiteness that these materials are plastic or semiplastic (coal), under different ranges of pressures. Potash salt is plastic under pressures exceeding 2000 lb. per sq. in. The bearing strength of salt is over 3000 lb. per sq. in. Also, it is important to know, if a method of mining such as suggested by the writer to increase the recovery is adopted, how a salt roof will behave if its strength as a beam or a slab is exceeded. Some of the present rooms have 40-ft. spans. Naturally the mine management wishes to know if there is danger that the roof may shear and the shear extend upward to the water-bearing gravels near the surface.

Testing Models of Mine Structures.—The testing of models of engineering structures having proportions and strengths the same as those of actual structures has been successfully done, and it has been tried for mine excavations to demonstrate the mechanics of ground movement. This is an old method and undoubtedly has been educational, but it is very difficult to get proper proportions and the relative strengths of the specimens similar to those in mines on a basis of, say, 1000 to 1, and to duplicate the action of stresses in mining operations. It cannot be said that in the past, except for the working out of special details, such as determining the bearing strength and plasticity, the results have been generally acceptable.

P. B. Bucky designed and developed an entirely new method of making physical laboratory tests.[9] He used a centrifuge revolving at a predetermined velocity; and his objectives are a "scientific method for determining the proper span or shape of roof for safe and economic mining." He presented a paper in 1938, also.[10] These laboratory experiments were of great interest to the committee. The scale of comparison being of the order of 1300 to 1, great skill has to be used to construct the models with the proper ratios of strength of the natural minerals, rocks and supports of a mine. The successful application to a mine of the data obtained will be of wide interest.

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[8] C. A. Pierce: Convergence of Roof and Floor in the Mine of the United States Potash Company. A.I.M.E. *Tech. Pub.* 985 (*Min. Tech.* Nov. 1938).

[9] P. B. Bucky: Application of Principles of Similitude to Design of Mine Workings. *Trans.* A.I.M.E. (1934) **109**, 25–50.

[10] P. B. Bucky and R. V. Taborelli: Effects of Immediate Roof Thickness in Longwall Mining as Determined by Barodynamic Experiments. *Trans.* A.I.M.E. (1938) **130**, 314.

Instantaneous Outbursts of Gas

At first the subject of instantaneous outbursts of gas seemed very remote from ground movement and subsidence. However, it has been handled by this committee, and with good reasons for doing so.

In the writer's first important investigation on this subject, he was asked to report upon "bumps and outbursts of gas" in the Coal Creek mines, Fernie field, British Columbia, 1916. Both phenomena had occurred in those mines and it was believed that they were part of the same phenomenon. This, however, was found to be otherwise, except incidentally as discussed in Introductory Notes on the Origin of Instantaneous Outbursts of Gas in Certain Coal Mines of Europe and Western Canada.[11] Briefly the writer's views are that hydrocarbon gases given off in coalification of vegetable matter and subsequently with increase in rank by pressure and heat were compressed out of the coal and absorbed in certain places called by the Germans "nests." These places, having been formed by past geologic folding and sealed under the high compression of the surrounding stratum, the gases have not escaped or migrated to the outcrops or the surface, but have remained absorbed or held under pressure in the nests until mining has approached near enough for the rupture of the surrounding natural dam.

Where the outburst gas is carbon dioxide, as in the Gard Basin, France, and in the Lower Silesian coal fields, the carbon dioxide was probably derived from carboniferous limestone by the heat of igneous intrusion and escaped upward into the coal bed through more or less vertical fracture faults caused by the intrusions.

Some investigators in Europe, French and German, think the gaseous outbursts come from crushing of coal in place by sudden compressions caused by the method of mining, as for example, bumps. This view does not, however, account for the carbon dioxide outbursts in the Gard Basin, France and in Upper Silesian mines, Germany.

Fortunately, coal mines in the United States have not experienced instantaneous outbursts of gas in which violence is considered the characteristic, except to a limited extent in the anthracite district where there has been geologic folding, but so far as the writer has obtained information there is nothing parallel in the United States to the violence of methane outbursts, like that of a local explosion, such as have occurred in one mine on Vancouver Island, a few mines in the Crow's Nest Pass district, British Columbia; in certain mines in Belgium; South Wales anthracite mines and of carbon dioxide gas in mines of the Gard Basin, France, and Upper Silesia.

The problem is not so easy of solution, but probably the best means is that now employed in Belgium and Upper Silesia of dividing the mine

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[11] *Trans.* A.I.M.E. (1931) **94**, 75.

into operating districts and when there are indications of high-pressure gas as in "blowers," at the coal face, the method is to cause ground shock waves by firing, after withdrawal of men, heavy charges of explosives in a large number of shots, along the face. These are fired simultaneously by electric switch at the surface or in a distant protected refuge place, equipped with strong doors and provided with compressed-air lines and oxygen-breathing apparatus. While as yet the United States has not experienced such outbursts, there is always the possibility that in the Rocky Mountain and Pacific Coast coal fields where they are deep and subject to volcanic effects there may be geologic conditions to produce them.

Summary of Suggestions for Further Studies of the Ground Movement and Subsidence Committee

1. Continued effort to obtain precise data on surface subsidence resulting from any kind of mining accompanied by: (*a*) time studies giving successive positions of face with relation to advance of subsidence; (*b*) "angles of draw" in specific cases and relation to geologic conditions; (*c*) ratio of vertical amplitude of subsidence to thickness or height of excavation of mineral mined; (*d*) effect of back-filling of different kinds; (*e*) ratio of amplitude of maximum subsidence to depth below the surface of excavation and with relation to time interval; (*f*) effect of dip or pitch of workings on surface subsidence; (*g*) lateral movement or "creep" of surface adjacent to subsidence area.

2. Continued effort to obtain data on underground movements in all kinds of mining: (*a*) deformation of pillars; (*b*) convergence of roof, or hanging wall; also time relation to different stages of convergence.

3. Extent to which excavation in the mineral being mined causes local roof pressures adjacent to working faces: (*a*) In the working place and goaf, to be determined by groups of props having hydraulic pressure-recording and time-recording devices, or else by special props, the compression of which under different loads can be measured at regular intervals, their compressibility having been previously calibrated in testing machines; (*b*) to determine as far as practicable at points at regular intervals in headings ahead of the face, the compression on the solid mineral. This has special application to coal mining, but might be applied to other kinds of mining as far as practicable under the conditions of formation and methods of mining.

4. Special studies of the effects of the geologic structure on ground movements and subsidences in which the Committee should have the assistance of the Committee on Geology in determining the character of the rocks, the presence and effects of faulting and folds, of dikes and intrusions. For coal, the effect underground of slips and faces are well known, but the presence of joint planes in the overlying rocks is not always

recognized and may not always have the same effect as the faces or joints of the coal bed.

5. Further, in respect to subsidence and ground movements from coal mining, practically all the data so far gathered by the committee for coal mining is in reference to flat or nearly flat coal beds. It is desirable to get similar data in coal beds that are dipping at various degrees.

6. More specific information is needed on bumps in coal mines and rock bursts in deep metal mines, especially as concerns related geologic structure and characteristics of rock strata; also, scientific indications of pending bumps and rock bursts.

7. The effect of subsidence on surface structures, and where some subsidence is inevitable, to plan the construction of buildings to minimize damage and designed, as in some mining villages in France, for raising by jackscrews as the foundation sinks.

8. Recording the findings of legal decisions in subsidence damage suits.

Collapsible Steel Props in Longwall Anthracite Mining

By John W. Buch,* Member A.I.M.E.

(New York Meeting, February 1939)

Nearly 25 years ago operating officials in the northern anthracite field were confronted with the problem of profitably mining virgin beds of thin coal (those 48 in. and under) or destroying them by mining underlying thick beds. To solve this problem costly hand-mining methods were replaced by cheaper mechanical methods, which eliminated the need for the lifting of bottom or the brushing of top in excessive quantities, which was essential if proper height for hand loading[1] was to be provided.

During this period long-face mining was an outstanding development. In its favor in comparison with any other method were the ease with which undercutting machines could be used, a differential in powder cost of about fifteen cents per ton in favor of this method over mining from a solid chamber or pillar face, and a greater production per day from a single working place, resulting in lower capital expenditures for a given total production, closer supervision and lower transportation cost.

The long faces were drilled by jackhammer and undercut by machines. The coal from the cut was loaded into mine cars placed on the gangway road by means of a double-drum mechanical scraper engine and a mechanical V-type scoop. A tugger hoist was used to spot empty cars at the loading point from a supply placed at the operation by an electric mine locomotive.

During loading the roof was supported by vertical wooden props paralleling the face. After the loading was completed the scraper engine and the ropes that had been used to propel the scoop in the loading of coal transferred the props to and along the face. A row of unfilled cogs was then built parallel to the new face and about 6 ft. from it. These cogs were usually made 6 by 3 ft., the longer dimension paralleling the face; they were placed on an average at 12-ft. centers. These wooden cogs were left in place and were buried later by falling roof. Attempts to remove them were never successful. When a new long face was started, the cog spacing would be increased to about 20-ft. centers. As successive cuts were made, weight would begin on the cogs, and they

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[1] References are at end of paper.

would gradually crush as the long face advanced, but they would not let the roof cave properly and thus relieve the weight on the working face. When the roof began to show this pressure, the cog centers would be gradually decreased from day to day until the cogs were virtually placed one against another on about 8-ft. centers. This did not serve to relieve the roof action, but rather to aggravate it, yet the practice was continued to protect the workmen against falls of broken roof rock. The more cogs built, the less chance there was that a free fall of roof in the mined-out area would relieve this weight along the working face. Finally, after the long face had advanced 200 or 300 ft., it often caved and was lost, or the roof became unsafe and the long face unworkable. A new face would then be made by reopening along the edge of the cave or by driving two chambers.

This method of roof control introduced seemingly unnecessary hazards, was expensive because of timbering cost and the need from time to time of re-establishing new long faces. Also, it interfered with production from the beginning of roof movement until the long face caved or was abandoned. However, the factors in its favor, as previously mentioned, were of such weight that for nearly 15 years it was preferred to the chamber-and-pillar method for mechanical mining in thin beds, where such a choice had to be made.

Steel Props Introduced

During the latter part of the 1920 decade, much effort had been expended in improving this phase of mining, and the importance of mechanical production became more apparent to both bituminous and anthracite operators. In conjunction with long-face mining in the bituminous field of central Pennsylvania a collapsible steep prop, which became known as the "Langham" prop, was developed, the use of which successfully induced caving near the working face and thus relieved the latter of the destructive face weight mentioned.[2,3] Since the introduction of these props late in 1929, one operator with their aid has mined more than two and one-half million tons of coal from longwalls, using them on seven longwall faces.

The Langham collapsible steel prop, shown in Fig. 1, is a rigid steel prop, capable of withstanding a center load of more than 450 tons. This strength has effectively supported the roof along working faces when adjacent mined-out areas were caving. Once established, an artificial break line is maintained by moving the props forward as the face advances, resetting them on a line parallel thereto. This prop, as its name denotes, is a truly collapsible prop. It is of the wedge-and-key type, designed to collapse instantly from its full height to a height of about one foot. It is made of cast semisteel, in three sections—top, middle and bottom. The middle section is merely a wedge-shaped piece of solid steel of the regular

prop diameter. Attached to it is a key held in place by the nut of a horizontal bolt, which passes through the wedge. When the prop is set for use, the key is turned by hand to an upright position, holding the separate pieces in place. When a collapse of the prop is desired, the key is turned to a horizontal position, using a 6-lb. hammer with a 4-ft. handle. To keep the three prop members from flying when the prop is released, and to aid in their removal from under fallen roof rock, they are loosely connected by a small chain. It has been found that a 5-ft. prop having a total weight of 422 lb. is the longest unit of this design that can

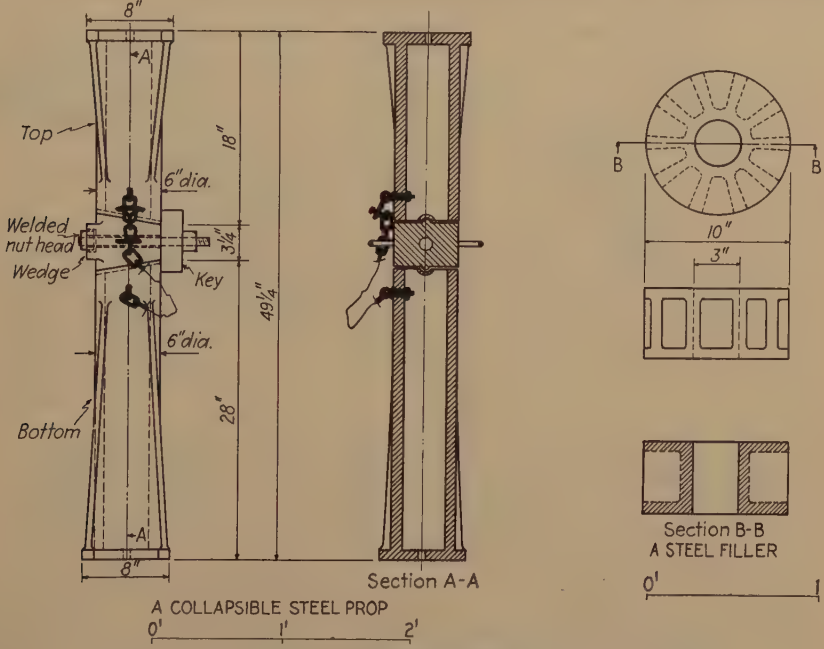

FIG. 1.—LANGHAM COLLAPSIBLE STEEL PROP AND FILLER.
Left, prop; right, filler.

be handled successfully in actual work. Where roof heights greater than this plus the 3 to 6 in. for the necessary wooden blocking are encountered, cast semisteel filler pieces 5 in. high and 10 in. in diameter are placed on top of the prop to build it up to the required height. When put into position for use, such filler pieces are always separated by 1-in. hardwood boards 10 in. square, to aid in the uniform distribution of weight over the metal surfaces. The face angle of the wedge is about 10.5°, and is exactly correct for a safe collapse under full load. Although various lengths of prop members can be purchased, it has been found desirable to use top and bottom members of nearly equal length, especially with total prop lengths over 3 ft., as this combination induces the most satisfactory collapsing action.

Paralleling the introduction of the Langham prop was that of a European type, consisting of two pieces of 4-in. steel H-beams, separated near the top of the prop. The design permitted about a 4-in. collapse, which frequently would not be sufficient to permit their removal without blasting, and this soon made them unserviceable.

Practice with Scraper Loader

The shaking conveyor has for the most part replaced the scraper loader on longwall work in this field. Conveyors are better suited to the large tonnages obtainable from long faces and with them there are no steel ropes to damage the steel props. A description of the use of

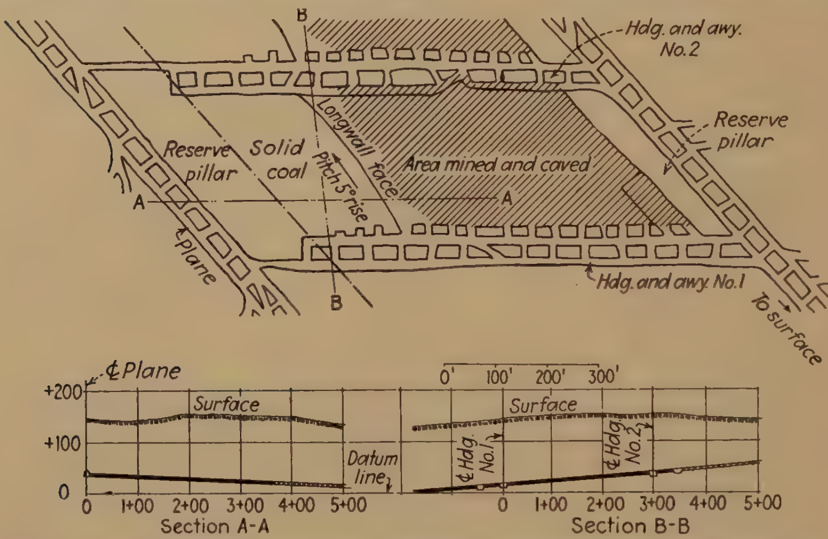

Fig. 2.—Plan of mine workings, showing advancing longwall face using scraper loader and collapsible steel props.

shaking conveyors is thought advisable, however, because it may become desirable in the mining of beds of coal so thin that hand loading into conveyors would be difficult, and also because the first experimental work with these props employed a scraper loader. This description is of an experimental operation from which more than ten thousand tons of coal were mined over a period of a few months.

The section of the mine where this work was done is shown in Fig. 2. A block of coal had been prepared in the usual way for advancing long-face mining. Stumps of coal were left along the haulageway until the mining of the block was completed and were then removed by hand. The bed was 35 in. thick, had uniform contours, and lay nearly level. The stratum immediately above the coal measures was a fairly compact shale rock about 3 ft. thick. Above this there was 122 ft. of cover, consisting of 85 ft. of sandstone and 37 ft. of surface wash, a total cover of 125 ft.

(Fig. 6*a*). The rock underlying the bed was of medium hardness. Steel props 26 in. long, weighing 172 lb., were used, permitting the use of 4-in. wood cushions at top and bottom. This wooden blocking was recovered and re-used on an average four times. Two double rows of steel props were set in staggered, alternate pairs, on 4-ft. centers parallel to the face, each pair of props footed and capped with 4 by 10-in. hardwood 42 in. long (Fig. 3). This is equivalent to one prop per foot of face. A 6-ft. pullway for the scraper was left open between the face and the first row of props. The lateral spacing between the two double rows was about 30 in., so that the 6-ft. pullway could be maintained on moving up

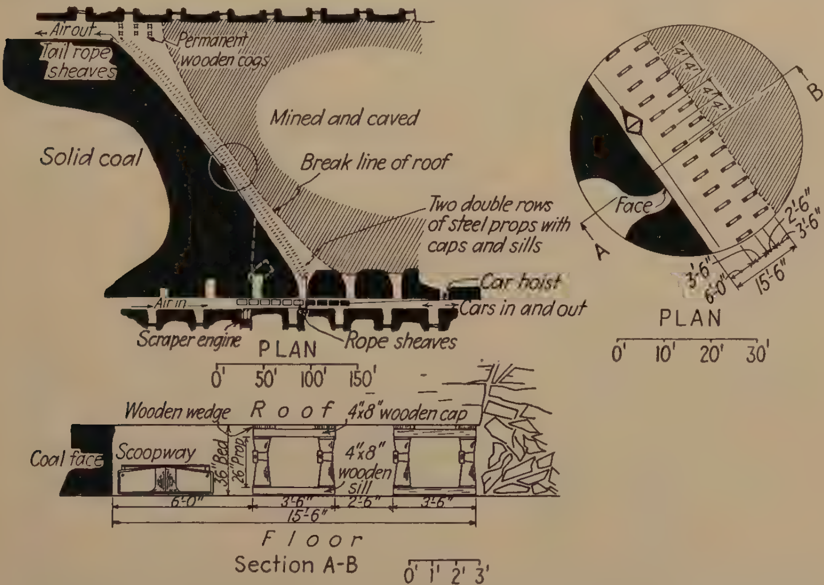

FIG. 3.—DETAILS OF LONGWALL FACE USING SCRAPER LOADER AND STEEL PROPS.

the double row of props nearest the gob after the loading of each cut of coal was completed. While the shortest distance between the loading gangway and the airway at the top of the wall was 300 ft., the length of the curved face essential to the satisfactory operation of the scraper was about 350 ft. A second opening to the airway above was maintained at all times.

The cycle of work was accomplished on 200 lin. ft. of this face in 16 hr. Coal was loaded on the 7 a.m. shift. Six men engaged in this work were classified as follows: one contract miner, who acted as face boss; two scoopmen; two carmen; one scraper engineer.

By 10 a.m. a path had been cleaned by the scoop from the front to the back of the cut and about 30 ft. of the face at the back had been entirely cleaned and all coal from this point loaded. At this time (10 a.m.) three propmen began their day's work. These men were

classified as follows: one miner, who had charge of the setting of props and who actually removed them; two laborers, who assisted the miner and whose work consisted largely of moving the steel props to their new position and resetting them.

These men proceeded at once to remove and reset props at the back of the cut where it was now free from coal and from which point the loaders had moved away. Roof falls in no way interfered with the work of loading coal. Loading was completed at 3 p.m. (8 hr.). The prop moving was finished by 6 p.m. (8 hr.).

The cutting shift reported for work at 3 p.m. There were three men on this shift, as follows: one miner, who had charge of the shift and drilled the face; one machine runner, who operated the coal-cutting machine; one helper, who assisted either the miner or machine runner.

These men drilled, cut and blasted enough coal for the succeeding days loading and completed their work by 11 p.m. (8 hr.). The low capacity of the scraper limited the length of face to about 200 lin. ft. While disadvantageous from a production standpoint, the 200-ft. cut was an advantage in maintaining the face curvature, so necessary to keep the scraper in contact with the solid coal face. The operation continuously averaged 115 tons of 2000 lb. each daily, or about 9.6 tons per man-shift.

In starting the operation, the roof was disturbed very little until the face had advanced 50 ft. At this point the "blackrock" immediately overlying the coal fell. After a total advance of about 125 ft., a break occurred in the main roof, which continued through to the surface as the face advanced. The rate of advance was 6 ft. per day (24 hr.), excluding various interruptions in working time because of slack market. Falls of main roof caused no noticeable disturbance underground. Often the "blackrock" stood without caving for several days after the props were removed, showing that there was very little pressure on the working face from hanging main roof. The breakline made its nearest approach to the props at the center of the long face. The face, however, was never lost while this block of coal was being mined. In idle periods of two or three days duration no change in roof pressure was noted in the working place and the roof did not settle enough to create any difficulty. One prop was lost in a cave. As will be seen later, this location was ideal for the use of the Langham props; they supported the roof without any structural failure.

Practice with Shaking Conveyors

The retreating longwall method employing collapsible steel props is being applied to the mining of a 165-acre tract of coal land where there

are two beds of coal yet to be mined, lying nearly level and separated by 20 to 40 ft. of sandstone strata. These are known as the Dunmore No. 3 bed (6 ft. thick) and the Dunmore No. 4 bed (4 ft. thick). As shown in Fig. 6*b*, the average total cover overlying the Dunmore No. 4,

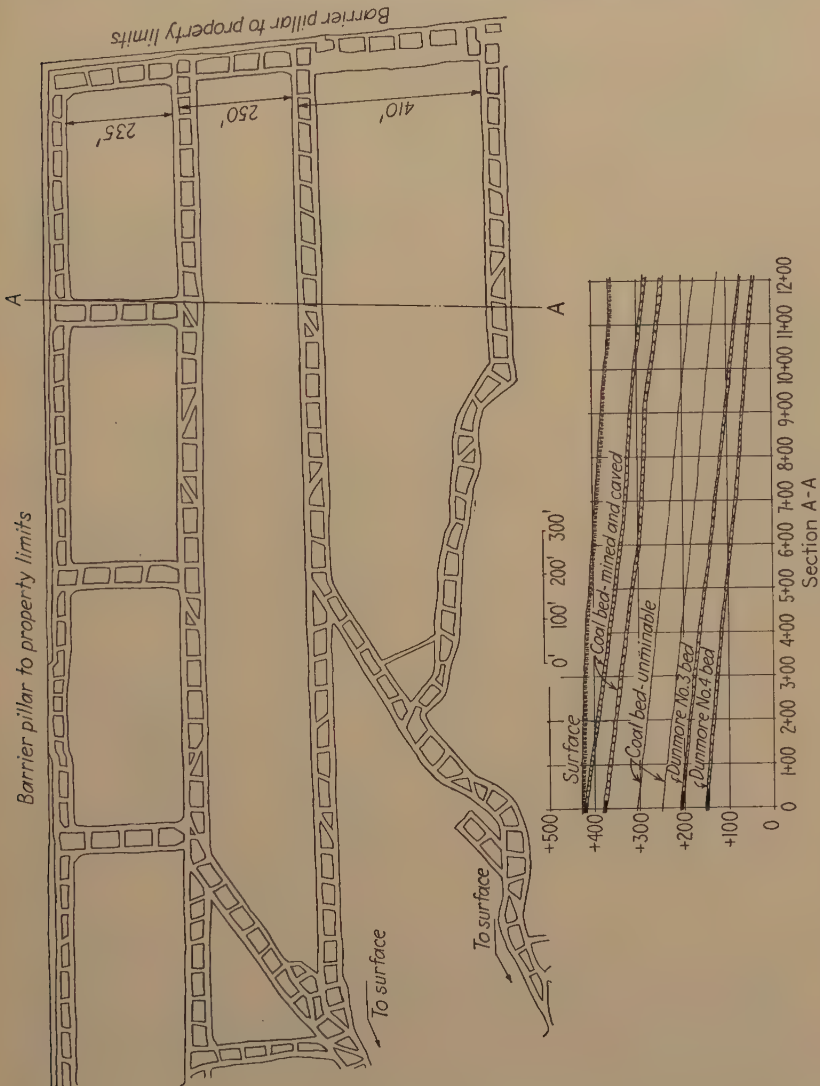

FIG. 4.—PLAN OF MINE WORKINGS USING SHAKING CONVEYOR AND COLLAPSIBLE STEEL PROPS.

which is the lower bed, is 304 ft. Two other beds nearer the surface have been mined out completely, thus fracturing the top 106 ft. of cover. Separation rock, 150 ft. thick, overlying the Dunmore No. 3 is intact. As, in the sequence of mining, that bed is being removed before the

Dunmore No. 4, a broken dead weight of 304 ft. is encountered when Dunmore No. 4 bed is to be extracted. Mining of both of these beds has subjected these props to an extreme test, but operator and manufacturer have met this test by redesigning and strengthening all parts of the prop, so that now difficulties from prop failures under this extreme loading are minimized.

The plan of mining in the two beds is similar. The general scheme of development (Fig. 4) is like that already described, though conveyors are used instead of scrapers. Longwall faces are mined retreating. When the first block had been defined in the upper bed adjacent to the property lines, mining was started to prepare a longwall face by driving two chambers at the extreme end in coal only, the outby chamber rib serving as the face for the longwall. The sequence is such that by the time this face has retreated a reasonable distance, say 300 to 500 ft., a similar face is ready in the next adjacent block below; and so on, a similar sequence of mining being followed in the underlying bed as the upper bed is mined out. The 30-ft. gangway chain pillars are left intact until mined as a part of the next adjacent longwall face, thus providing a means of ventilation and a second opening. No stump pillars are left in the longwall block. Steel props are used for temporary support along the gangway, so as to provide space for 15 empty cars, and these props are removed as the mining retreats.

The following description refers to the practice of mining the Dunmore No. 3 bed. Included in the 6-ft. average bed thickness is a top band of slate 18 in. thick, termed "blackhead," which falls or is shot down as the coal is mined, and which must be gobbed or loaded. The roof above the blackhead is compact sandstone. The bottom rock is smooth compact slate. Coal is shoveled by hand into a shaking conveyor of 70 tons per hour capacity. The face arrangement is shown in Fig. 5. The conveyor drive is installed halfway up the wall, so as to equalize the load on the pan connections. A clearance of 40 in. is provided between the conveyor and the solid face for the travel of a longwall coal-cutting machine. Two double rows of 5-ft. steel props are used on the gob side of the conveyor, placed on 30-in. centers paralleling the face in staggered, alternate pairs. This is a variation from the 48-in. spacing described in connection with the scraper mining and is made to reduce the load on the separate props because of the heavier roof loads encountered. The props are tied together in pairs by placing on their tops hardwood cap pieces 2 in. thick, 10 in. wide and 42 in. long. The space remaining between this tie piece and the roof is filled by building up with 5-in. steel fillers and with wood blocks 10 in. square and of various thicknesses up to 4 in. The props are set on hardwood blocks, 10 in. square and 4 in. thick. This wooden blocking material is removed and re-used. Lateral spacing between these pairs is about 30 in., sufficient to maintain the

proper relative positions on moving up the double gob row of props with each successive 6-ft. cut on the face. The props are placed so that their wedge bolts are at an angle of about 15° with the pan line, the wide part of the wedge facing the gob and up the wall. It has been learned that this manner of pointing the bolt and wedge is safest because it minimizes the hazard of flying prop parts when collapsing them or of flying bolts or wedges in case of prop failure. Further, it is the most satisfactory position for approach in removing them. A temporary single row of steel or wooden props, the number dependent upon the condition of the roof

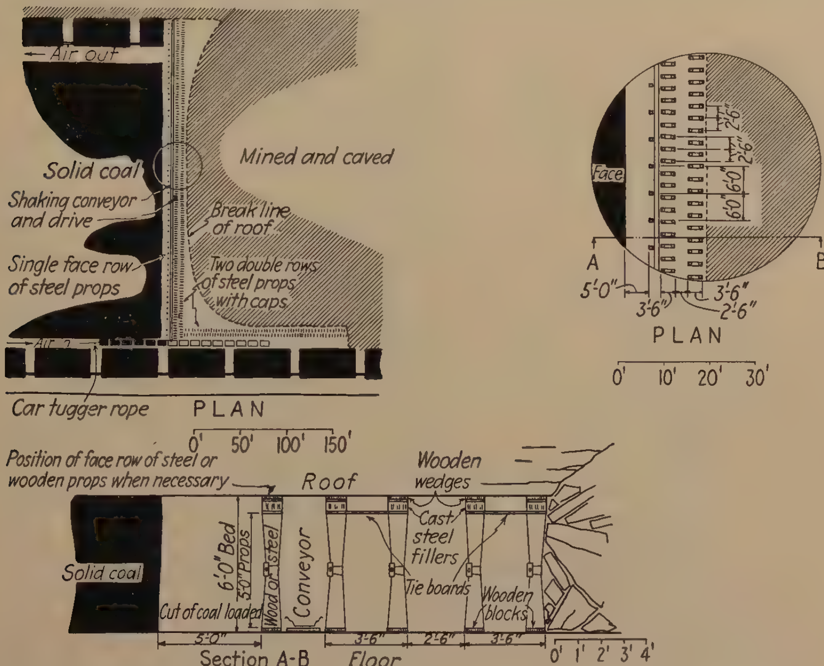

FIG. 5.—DETAILS OF LONGWALL FACE USING SHAKING CONVEYOR AND STEEL PROPS.

encountered, is placed between the conveyor and the solid face during the loading of coal and removed when the conveyor pans are moved up if conditions permit. If the roof is so bad that some of these props cannot be removed, the pan line is disconnected for moving past them, the removal of the steel props being accomplished when they come into a position near the gob line two cuts later. When wood props are so used they are not removed. At the loading point a tugger hoist moves car trips in front of the conveyor. Jackhammers drill the face, which is blasted with pellet powder.

In describing the cycle of work, assumption is made that a full cut of coal is ready for the loading shift, which starts at 7 a.m. and completes

the work of loading at 2:30 p.m. (7 hr.). This crew consists of one face boss; one laborer for each 30 ft. of face, to load the coal from that section of the wall and set the necessary face props; two carmen, who see that the cars are placed and properly loaded, and who operate the control switch for the conveyor. The work of moving the conveyor, moving props, undercutting and drilling the face is done on the 3 p.m. shift in 7 hr. The men on this shift are divided into two crews, one for cutting and drilling the coal face, the other for moving the conveyor and props.

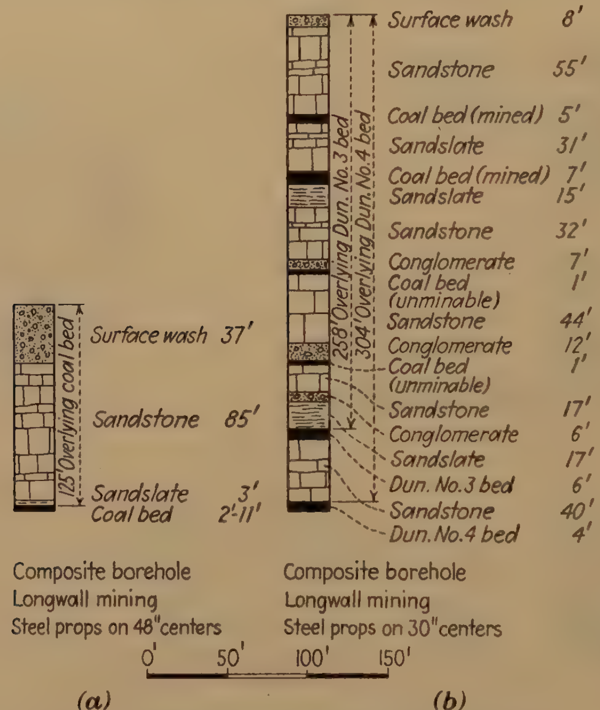

FIG. 6.—COMPOSITE BOREHOLE, LONGWALL MINING.
a, steel props on 48-inch centers.
b, steel props on 30-inch centers.

The cutting and drilling crew consists of five men: that is, one miner, two drillers and two machine runners. The moving crew is composed of 12 men, as follows: one miner, who supervises the work; three laborers, who move the conveyor; eight propmen, who work in groups of two men each for each 60 props to be moved.

Blasting is done on the 11 p.m. shift; then the crew trims and gobs most of the blackhead, which comes with the cut, and, in general, finishes preparing the face for the succeeding day's loading. This crew consists of six men—one miner and five laborers.

In all, 37 men are required, divided as follows:

7 A.M. Shift	3 P.M. Shift	11 P.M. Shift
1 miner	1 miner	1 miner
12 shovelers	2 drillers	5 gobbers
2 carmen	2 machine runners	
	8 propmen	
	3 conveyor movers	
15	16	6

The average performance for a complete cycle of 24 hr. for one such operation as has been described, covering a 3-yr. period and adjusted to conform to the 7-hr. day, is given below:

Mine Cars Loaded		Number 2000-lb. Tons Coal Produced	Average Number Shovelers	Average Total Number Men	2000-lb. Tons of Coal per Shoveler	2000-lb. Tons Coal per Man
Coal	Blackhead					
150	12	470	12	37	39.2	12.7

Roof Action

On starting a longwall face a retreat of about 300 ft. is usually made before breaks occur in the main roof over these seams. Just previous to such breaks the props are subjected to an excessive weight, which causes a few of them to fail. After the first main roof break occurs, the roof continues to break with every face retreat of 70 to 80 ft., the break finally reaching the surface. The roof immediately overlying the coal bed falls almost daily at the time or soon after the props are removed. Near the center of the wall, the break line is against the props, and gradually recedes toward the mined-out area until it is about 40 ft. from the prop line at both ends of the wall. This arc-shaped break line is attributed partly to the steel props used to maintain car space along the gangway road at the lower end of the wall and partly to the fact that the cave along the face is slow to join the cave in the mined-out area above the wall. The arc-shaped break line prevails in both Dunmore No. 3 and Dunmore No. 4 bed. At no time has the roof weight encroached over the props, and the coal face has never been subjected to noticeable pressure by overhanging main roof. As soon as they are collapsed, the props are reset in their new position; which insures proper roof support at all times with four rows of steel props.

Safety

At one mine of which the production is nearly all from longwalls, where shaking conveyors and steel props have been used, the record over a period of 6½ yr. shows that immediately after this method was

TABLE 1.—*Injuries Chargeable Directly to Use of Steel Props*

Causes	1932		1933		1934		1935		1936		1937		First Six Months of 1938	
	Number	Man-days Lost	Number	Man-days Lost	Number	Man-days Lost	Number	Man-days Lost	Number	Man-days Lost	Number	Man-days Lost	Number	Man-days Lost
Parts of props flying	10	964	5	232	1	13	2	9	0	0	0	0	0	0
Handling props	11	230	6	60	5	148	5	72	1	10	0	0	0	0
Props collapsing	2	149	0	0	2	126	2	103	2	18	0	0	0	0
Bumping into prop	3	39	0	0	0	0	0	0	0	0	0	0	0	0
Totals	26	1,382	11	292	8	287	9	184	3	28	0	0	0	0
Totals on a per day basis	0.19	9.9	0.05	1.3	0.03	1.2	0.05	0.9	0.01	0.13	0	0	0	0

introduced a number of injuries occurred that were directly chargeable to the use of the props, either in the handling of the prop members or from parts of the props flying on failure and striking men. Changes in design and in setting position reduced the number of injuries from year to year until, during the year 1937 and for the first six months of the year 1938, no injuries chargeable to steep props were incurred. Table 1 shows this in detail.

This mine has not had a roof-fall fatality since the introduction of steel props. In carrying out a company policy of acknowledging quarterly the best safety record among its 20 or more mine foremen, by the award of a safety banner, the mine foreman in charge here has received this award six of the thirty-two times it has been presented. The next best record is five times, at the mine where the steel props were first introduced for experimental purposes and where later much use was made of them until the mine was finished about two years ago. Table 2 gives the safety record of the mine with the conveyor system discussed in this paper, covering a period of 5½ years.

TABLE 2.—*Safety Record of Mine Using Conveyors and Steel Props*

Year	Lost-time Injuries	Man-days Lost	Man-hours Exposure	Frequency	Severity	Fatals
1933	47	7,779	526,373	89.29	14.78	2
1934	53	4,481	621,097	85.33	7.21	1
1935	48	4,161	546,268	87.87	7.62	1
1936	33	1,421	559,377	58.99	2.54	
1937	30	1,294	442,017	67.87	2.93	
First Six Months 1938	11	147	182,894	60.14	0.80	
Total	222	19,283	2,878,026	77.14	6.70	4

$$\text{Frequency} = \frac{\text{Number of Lost-time Injuries}}{\text{Man-hours of Exposure}} \times 1{,}000{,}000.$$

$$\text{Severity} = \frac{\text{Number of Man-days Lost}}{\text{Man-hours of Exposure}} \times 1{,}000.$$

ECONOMY

The discussion herein considers only timber-expenditure economy with this method and includes cost of props broken, props lost in caves, fillers of steel or of wood lost or broken and wooden wedges used to tighten the props, in comparison with timbering costs in long-face mining under comparable conditions operating with only wooden props and cogs. The average cost for roof support at this mine for three longwalls over a 5-yr. period was \$0.133 per ton of 2000 lb. The divisions of this cost are shown in Table 3.

With long-face mining employing permanent wooden props and cogs for supporting the roof, the cost varied between \$0.400 and \$0.600 per

TABLE 3.—*Average Cost of Roof Support*

Material	Number	Cost per Ton
Steel prop bases and tops lost or broken	3,201	$0.055
Steel prop wedges, bolts and keys	11,937	$0.017
Wooden wedge material	907,583	$0.036
Steel filler pieces	8,496	$0.025
Total over five-year period		$0.133

ton. The economy, therefore, is between $0.267 and $0.467 per ton. Such an economy merits consideration and when added to reductions in other items of cost accompanying concentrated uniform mechanical production leaves no doubt that the method is a desirable one.

ACKNOWLEDGMENT

The author wishes to acknowledge the generosity of The Hudson Coal Co., in whose mines much of this work was done, for its cooperation in supplying data for this paper; also, the help of Mr. Harry Weaver, who assembled much of the information in preparation of a thesis for his professional degree of Engineer of Mines at The Pennsylvania State College.

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DISCUSSION

(*W. H. Lesser presiding*)

C. ENZIAN,* Hazleton, Pa.—It is gratifying to note that the highly satisfactory results were possible in the slow-yielding rock formation overlying the anthracite beds, without the use of extensive cribbing that was considered necessary during the early trials.

H. H. OTTO,† Scranton, Pa.—In 1930 the company I am with was experiencing trouble at one of its collieries, with steel props breaking. A number of props and some wooden cap pieces were taken to Lehigh University and tested at the Fritz Engineering Laboratory. The props stood a total pressure of 906,000 lb. The application of the pressure noted was not sufficient to cause a collapse of the props tested, but the center wedges of the steel props, in several instances, showed strain, as indicated by small cracks which appeared under the maximum pressure applied. The results of these tests, together with the actual failure of some of the props in use at one of our longwall operations, caused the manufacturer to re-design subsequent props, strengthening them at the point of noted failure.

The use of collapsible steel props is of great assistance in connection with longwall mining.

* Consulting Mining Engineer.
† Hudson Coal Co.

Development and Application of Concrete and Steel Roof Support Used on Haulageways, Pump Rooms, and Main Openings in the Anthracite Mines of Pennsylvania

By W. W. Wirth* and W. L. Dennen†

(New York Meeting, February 1940)

Research looking toward the reduction of the cost of roof support by substitution of longer-life materials for wooden timber is fully justified by the fact that roof support is an important element of all production costs. Permanent or semipermanent roof support will reduce the expense of timber-maintenance crews during steady operations, but even more markedly during idle periods. The management, therefore, is faced with the important problem of timber renewals, which in times of regular operation are frequently obscured by more urgent problems of production. Increased use of timber in the Anthracite Region of Pennsylvania is caused mainly by increased percentage of coal obtained from robbing areas, which requires more timber for safety than development work, and maintenance of long haulageways through crushed ground despite the fact that less coal is transported over them.

The timber reserve in the immediate vicinity of the mines has been depleted to such an extent that it is necessary to secure 70 per cent of the timber requirements from distant sources.

Originally, mine timbering in this region was done by largely utilizing round props or legs with round collars. The whole cross section of the tree from which timber was cut was utilized by forming the joints between the legs and the collars with saw and adze, using plank or round saplings for lagging behind the legs and over the collars to prevent material falling between them.

The Mine Inspectors' report of 1898 records an important digression from the use of round timber, when two tunnels at the Babylon colliery, near Pittston, Pa., were timbered with 12 by 12-in. square timber sets with mud sills, through quicksand for distances of 425 and 160 ft., respectively. The timbers were lagged with 3-in. plank to control sand.

An early step toward the permanent type of roof support was the installation of "brick pillars or perfect arched brick rooms" instead of

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wooden timbers in pump houses on the second and third levels of the Drifton No. 2 slope near Hazleton, Pa., as reported in the Mine Inspectors' report of 1899.

A similar illustration toward the permanent type of lining is at Plymouth No. 1 colliery (at Larksville), Bennett bed pump room, 17 by 59 ft. with 10 by 15 ft. offset, which was reinforced by stone side walls and brick top arches. The pumps were steam driven with condensers, which exhausted into the sump and caused a heat condition, the temperature ranging from 100° to 120°. This heat condition, in addition to an existent unstable condition of the bed itself, caused failure, and it was

Fig. 1.—Brick lining, foot of Pine Ridge shaft.

necessary to replace portions of it in 1919 with reinforced-concrete walls and steel I-beam collars. In due time the gradual movement and heat cracked the heavy reinforced-concrete side walls, bending the beams so that replacement was again required. In 1930 work was started on the installation of a concrete-block flexible lining, known as Schaefer lining, explained more in detail later in the text. The main pump-room lining was completed in 1931. Additional Schaefer lining has been added on both ends of the pump room, as well as on the suctionways. The steam pumps were replaced at this time by modern electric pumps, which eliminated the heat condition. The roof pressure of the area has now become stabilized and the Schaefer lining is proving satisfactory.

Brick was also used at numerous places in the gangways and shaft landings, and almost all of these replaced wood timber. An example is an arched brick lining to support the roof and side walls, which was installed at the foot of Pine Ridge shaft, near Wilkes-Barre, in 1903. This installation is still in an excellent state of preservation (Fig. 1).

Rectangular steel roof-support sets, manufactured by the Carnegie Steel Co., were first used in 1894 as a substitute for timber. One of the

early institutions of this form of support was in a pump room at No. 40 slope, Hazleton shaft colliery, replacing all the wood. In 1908, the Maxwell colliery, near Wilkes-Barre, protected a double-track gangway with 20-in., 65-lb. I-beam collars 17 ft. long between 8-in. H-beam legs 10 ft. 6 in. high in the clear, weighing with base plates 1720 lb. per set. These steel supports, erected on concrete bases, are still in use, having been in place about 30 years. They replaced yellow pine round timber 24 in. in diameter, weighing 5040 lb. per set, which experience found had an average life of 2½ yr. The timber replacements were compulsory, owing to deterioration and failure.

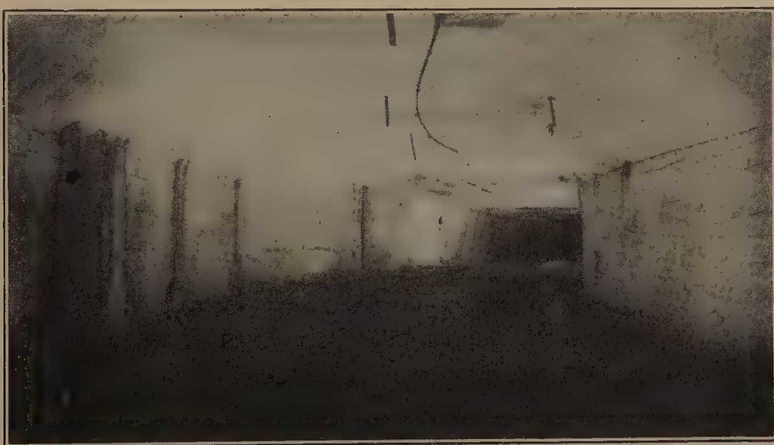


FIG. 2.—No. 5 SHAFT FOOT, BALTIMORE BED, BALTIMORE COLLIERY.

Where the life of the opening warranted its use, or where heavy roof pressure was noted, reinforced concrete came into considerable prominence about 1914. Fire protection was a secondary reason for the use of these materials. An example of this type of installation is Baltimore No. 5 shaft, Baltimore bed landing in North Wilkes-Barre, where a double-track shaft foot was reinforced on both sides and roof with reinforced concrete, about 1920, which replaced 16 to 18-in. round timber. This installation is still intact (Fig. 2).

In 1922 The Hudson Coal Co. experimented with Medusa precast concrete supports, installing three sets, replacing 18 to 20-in. round timber, in the Baltimore bed at Baltimore colliery (Fig. 3).

The bed at this location was originally 28 ft. thick, lying about 355 ft. below the surface. It was first mined many years ago, and about 60 per cent of the coal was removed, leaving about 40 per cent in irregular pillars, for support. The coal, being very free as well as slippy, has spalled off the pillars, causing heavy caves and crushing the pillars. The bed has now been crushed to about 12 to 15 ft. thickness. Some

idea of the roof pressure can be imagined by looking at the photograph (Fig. 3), which shows the precast concrete legs and collars in relation to the small wood timber sets in the background, which were necessary to keep this road open, as it is now used only as a drainage way in an idle section of the mine. This is a splendid example of concrete roof support of this type, which has been standing for 16 yr., in an extremely bad condition where the surrounding area is caved and inaccessible except for this portion of the roadway. These sets comprise flat reinforced-concrete collars, legs and spreaders or sprags with 4-in. reinforced-concrete slabs for lagging.

Fig. 3.—Medusa precast concrete supports near foot of No. 1 plane, Baltimore bed, Baltimore colliery.

During 1922, rectangular 15-in., 50-lb. steel I-beam support sets replaced 18 to 24-in. round timber sets on the Conyngham concentration road, Baltimore bed, Baltimore colliery. Fig. 4 shows a portion of this installation. The individual sets are spragged by means of a 4-in. pipe flattened on the ends and bent to form right angles, which are drilled and bolted in place. The bed in this area is about 26 ft. thick, and the voids above the sets are lagged with scrap rail or treated timber to the roof. This installation is also in first class condition.

A modification of this method is the use of reinforced-concrete side walls with steel I-beam collars. An example is No. 34 tunnel at Baltimore colliery, installed in 1928 (Fig. 5). This tunnel was driven from a double branch through caved ground. A 4-in. concrete mat was placed over the I-beams and the voids above were filled with silt. This installation is also in excellent condition.

In addition to pump rooms and haulageways as described, extensive installations of steel and reinforced concrete have been made to protect underground mule barns, locomotive pits, and other openings, all of

which gradually led up to the more modern installations of roof support, which (exclusive of wood) may be listed as follows: (1) concrete block

Fig. 4.—Rectangular steel support sets, Conyngham concentration road, Baltimore colliery, Baltimore bed.

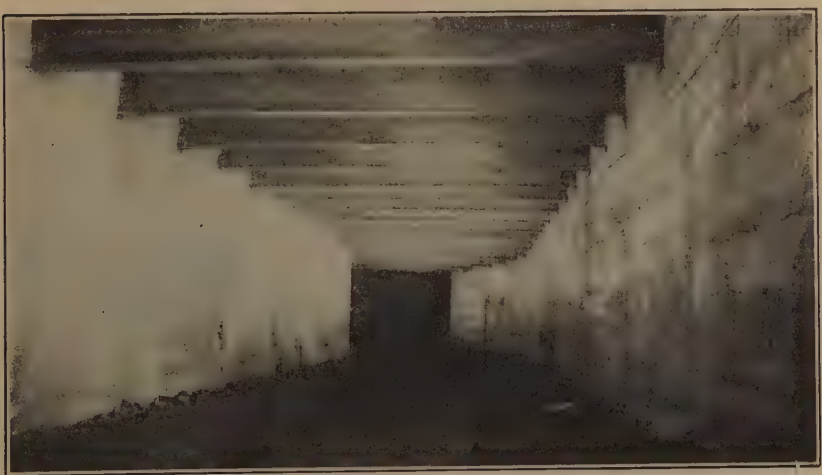


Fig. 5.—No. 34 tunnel, Baltimore bed, Baltimore colliery.

(Schaefer lining), (2) steel arches, (3) steel tunnel-liner plates, (4) precast concrete sets (legs and collars).

Schaefer Lining

The form of roof protection known as the Schaefer lining, using a patented process referred to previously, was designed and patented by Hans Schaefer, an engineer of Essen, Germany, and has been used for many years in Europe.

The first installation on this continent was in 1928, to protect a road where heavy timbers lasted only a few months because of squeezing of the overburden. Schaefer lining is designed to form a nonrigid, strong, secure arch of permanent fireproof material installed so as to yield slightly under pressure without breaking or collapsing, and thus permit or relieve

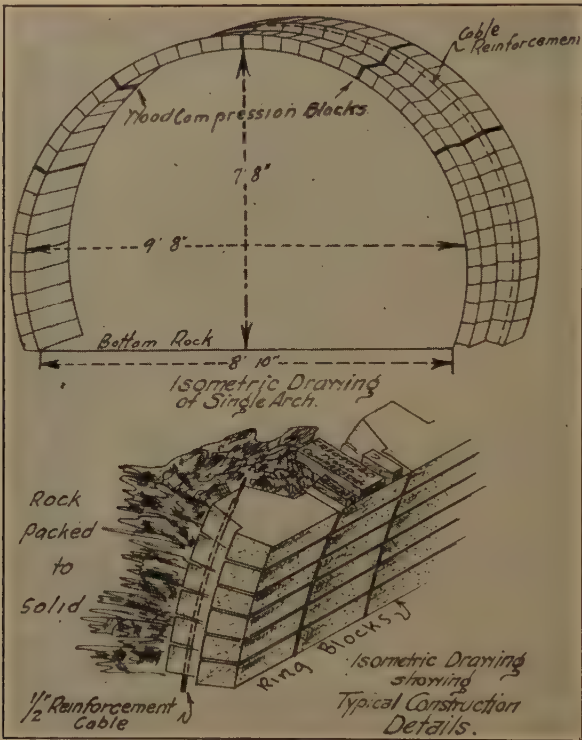


FIG. 6.—ISOMETRIC DRAWINGS OF SCHAEFER LINING.
Top, single arch; bottom, typical construction details.

all pressure from movement of heavy overlying masses of material. It enables the supported mass to settle until it becomes self-supporting, providing permanent maintenance of openings through crushed or broken ground. This lining consists of a series of individual arches erected skin to skin, using a special shape of concrete slab. The slabs are of uniform size, having a given arch with smooth surfaces. They are generally constructed at or near the mine where they are used. Each slab weighs about 75 lb., and can be handled by one man in the construction of the lining. The slabs are shaped roughly in the form of a T and are tapered or beveled to lay up naturally in the form of the arch of a given radius, the stem portion of the T-slabs containing a round hole

through which 5-ft. lengths of reinforcing steel are placed, connecting about 10 slabs around, in which grout is placed to protect the steel from water. This steel helps to hold each ring intact and tends to prevent distortion of the arch under heavy pressures.

These arches (Fig. 6) when laid up contain two to six rows of compression blocks of creosoted wood, to cushion the arch when pressure occurs and prevent breaking the courses. No mortar is used between the slabs of the arches nor between the separate arches in this construction. The arches are either circular or horseshoe in shape, seated upon a concrete or natural rock base, at the sides. In some places,

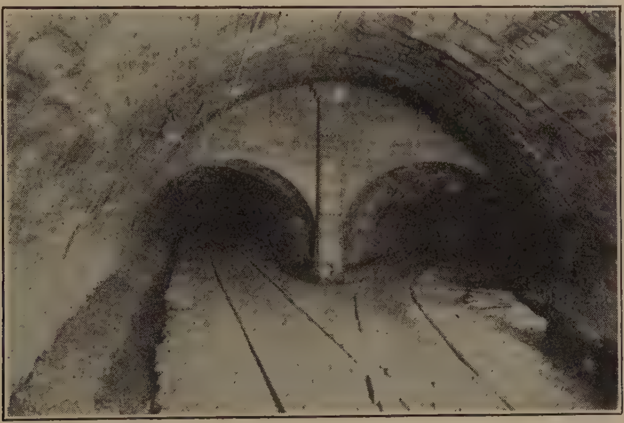


FIG. 7.—SCHAEFER LINING JUNCTION, POWDERLY COLLIERY.

where there is a soft bottom, or over filled ground, the main arch is erected upon an invert, the radius of which is approximately double that of the main arch, or in complete circles, and of similar construction.

A space is left outside these arches—not less than one foot throughout the perimeter thereof—which is packed tightly with fine material such as broken rock, silt or ashes, to cushion the pressure when squeezes bear upon the lining (Fig. 6).

The ingredients used in making up these blocks vary slightly, depending upon the conditions where the lining is used, and approximate one part cement, three parts clean sharp sand and one part crushed stone or pebbles of good quality not larger than will pass through a ⅝-in. mesh. In some places cinders are used instead of sand and gravel and these produce a very strong block. Cinder blocks are not used in wet conditions, but are more economical in dry conditions.

The first installation referred to above was at the Powderly colliery, near Carbondale, at the junction of two gangways, the one being a

single-track roadway 12 ft. wide and 6½ ft. high, the junction being 20 ft. wide and 12½ ft. high, as shown on Fig. 7. More recent junctions have been erected of variable widths, the maximum to date being 30 ft. wide.

The use of this material in Loree No. 1 pump room has been mentioned. This construction in 1931 protected a room 22 ft. wide and 75 ft. long, later followed by the construction of lining in the suction, discharge-ways, and approaches to a total of 750 lin. ft. of diameters varying from 8 to 22 feet.

Another installation of this form of lining to overcome a bad condition was in the Cooper bed landing of Loree No. 2 shaft, where the safety of the shaft was involved. This lining is 108 ft. long, ranging from 12 to 16 ft. in diameter (Fig. 8). The cost of this installation is given in Table 1.

TABLE 1.—*Cost of Lining at Cooper Bed Landing*

	Labor	Contract	Total
Schaefer lining, cinder blocks	$	$3,000.00	$3,000.00
Dead work, top rock, trimming ribs, etc		3,460.00	3,460.00
Recess and chimney for cable	13.00	40.00	53.00
Transportation and handling blocks, etc	241.49		241.49
Grand total	$254.49	$6,500.00	$6,754.49

Various pump rooms, engine rooms and gangways have been protected by this form of construction at The Hudson Coal Company's operations to a total of approximately 1500 lin. ft. of Schaefer lining of various diameters. An average cost of 10 ft. in diameter Schaefer lining of common single-track gangway is approximately as follows: Schaefer lining blocks, etc., cost per ring, $23.25; erection, 23⅓ man-hours @ $0.6609 = $15.42; transportation and handling materials, $0.90; total $39.57, or $71.23 per lin. yard.

Where there is a high percentage of acid in mine water, and where roof support is necessary in wet conditions such as water ditches or sumps, or where there are trickles of water heavily charged with acid, any form of concrete construction is subject to deterioration through the leaching of the lime in the cement. This leaching makes the concrete brittle and honeycombs it, and where it is reinforced exposes the steel to the action of the acid. When the base of the construction has been expected to be wet it has been made up of fire-clay brick with a suitable bond. Some minor replacements of Schaefer blocks have been found necessary and have been made under these circumstances. Where fire-clay bricks have been used in experimental work, they have been bonded together with a

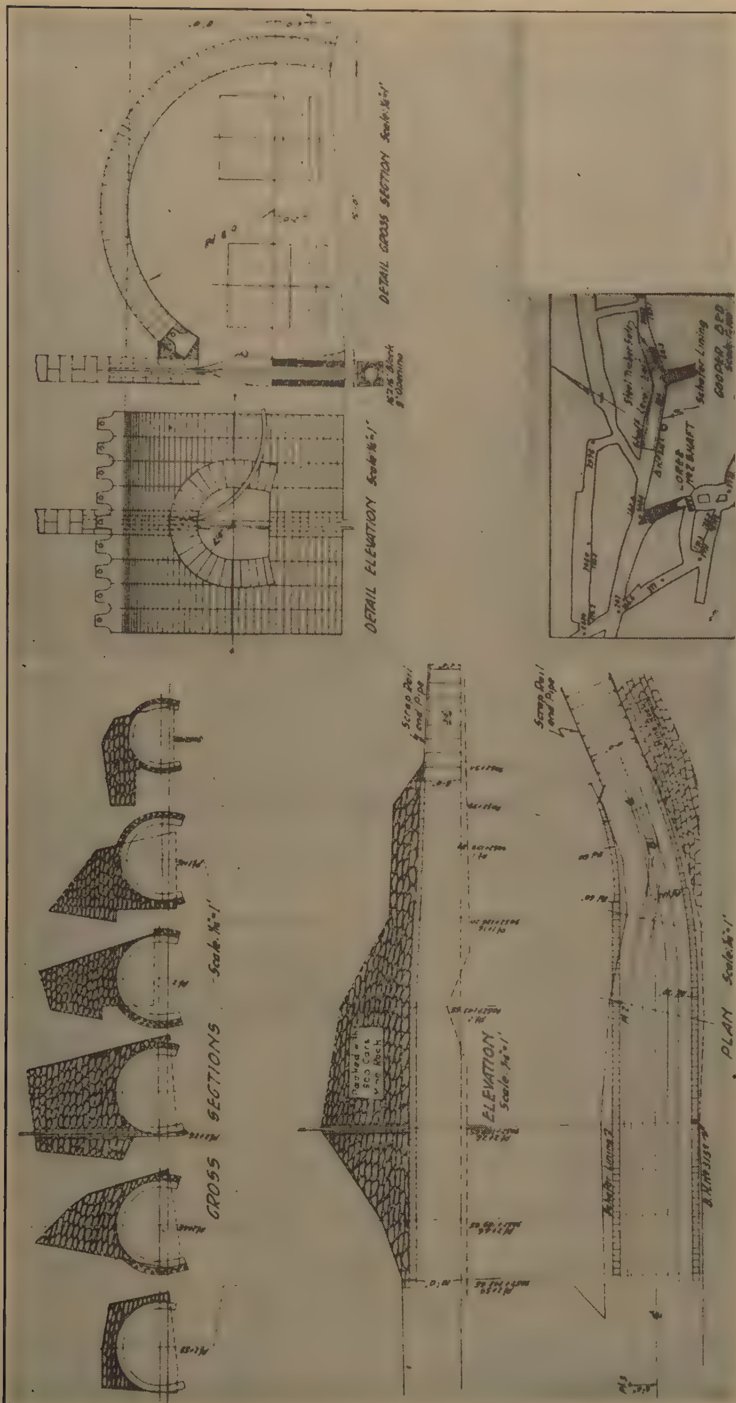


Fig. 8.—Schaefer lining, shaft level, East Cooper bed, Loree No. 2 shaft, Hudson Coal Company.

hot sulphur cement commonly known as Vitrobond. Another acidproof cement that has been used successfully is called Vitrex.

Where individual slabs of Schaefer lining have been damaged by acid-bearing trickles from the roof, they have been, as an experiment, replaced by a new part made up of fire-clay brick bound together with Vitrobond or Vitrex to make a finished block of the exact size of the concrete block replaced. The necessity for this repair work is very unusual, as the blocks

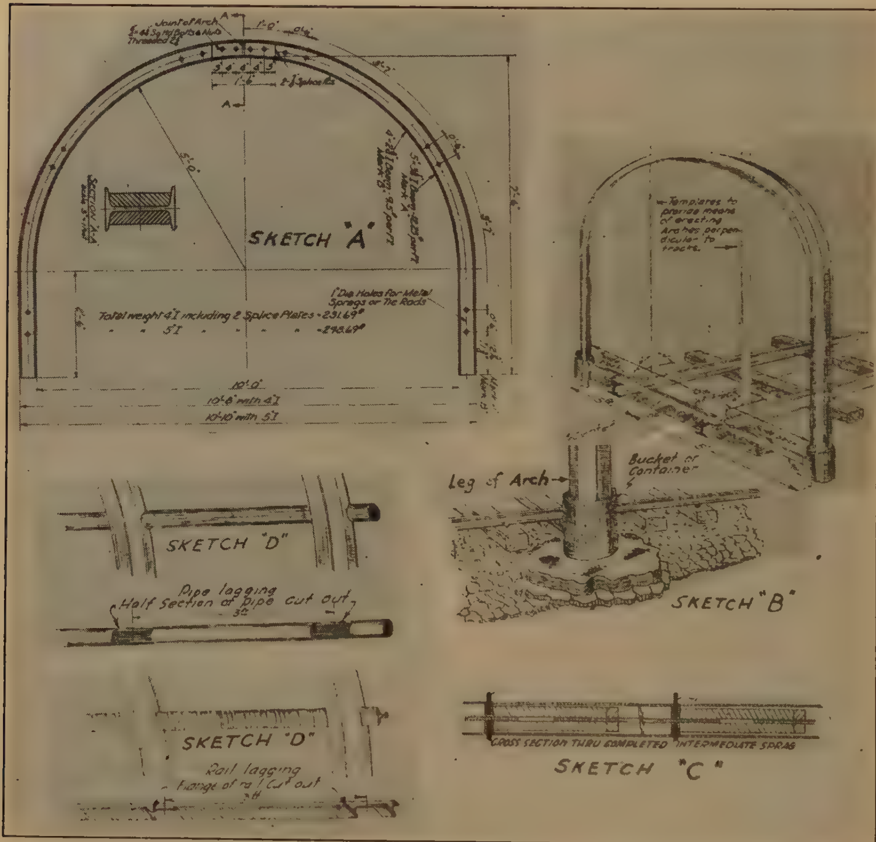


FIG. 9.—DETAILS AND CONSTRUCTION OF STEEL ARCHES.

or slabs that make up the arches described generally are very strong and withstand the wear and tear of all ordinary mining conditions.

STEEL ARCHES

Prefabricated steel arches, now approved as standard for use in certain mines, are shown in Fig. 9-*A*. They are of two different weights of material. The 4-in. curved I-beam weighs 9.5 lb. per lin. ft., while the 5-in. I-beam weighs 12.25 lb. per lin. ft. These arches are constructed by

bending 4 or 5-in. I-beams, giving a clear span of 10 ft. on a 5-ft. radius with a 2 ft. 6-in. leg. Each arch consists of two units, each embodying a 2-ft. 6-in. leg with a quadrant. The quadrants of each unit are connected at what becomes the crown of the arch by a ⅞-in. steel splice plate 18 in. long, held in place by ¾-in. steel bolts.

The application of each is determined by the conditions and depths below surface of the location where the installation is to be made. The steel arch type of roof support is used as a standard form of such support only at points where the use of so expensive a material is justified by the probable life of the opening, a minimum of which is considered to be 7 years.

Each of the officials having charge of the erection of this form of roof support is furnished with a Manual of Instructions, describing in detail the proper method of erecting, lagging and backfilling this steel construction.

The feet of the arches are set in small bucketlike forms (*B*, Fig. 9), upon 3-in. wooden blocks with sheet-steel plate cover on a solid rock floor or on a large slab of suitable rock as a footing and the "buckets" filled around the feet of the arch with a mixture of 50 per cent asphalt and 50 per cent sand, tamped solidly in place. The members of the arch are drilled at the factory to accommodate pipe sprags or struts (*C*, Fig. 9), spacing the units at the desired distance each from the other to prevent twisting. These struts are held by a ⅞-in. screw rod with a nut on each end, for convenience in bolting the struts into place. Sometimes sprags made of scrap rail or pipe (*D*, Fig. 9) are placed on the periphery of the arch. They are grooved to fit the arch sets at the colliery, and act not only as sprags but as lagging. The lagging may also be separate and consist of scrap shaker-chute pans, screen jackets, pipe, rail, et cetera.

TABLE 2.—*Average Installation Cost of Arch Set*

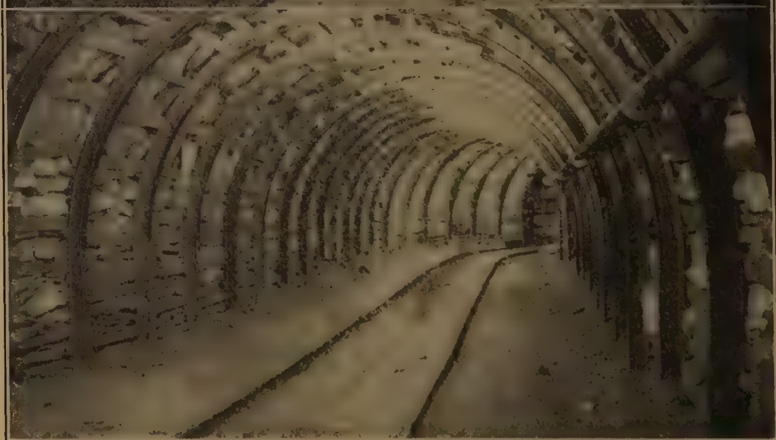
	Cost per Set
Material	
Steel arch support (4 in.)	$ 6.67
Scrap rail—sprags and lagging	2.70
Miscellaneous—cement, asphalt, tools, etc	0.80
Total material	$10.17
Labor	
Aligning and erecting arch supports, 16 man-hr. @ $0.6609	$10.57
Rock packing, 11 man-hr. @ $0.6609	7.27
Transportation and handling materials	0.70
Total labor	$18.54
Total labor and material	$28.71

Since 1933 The Hudson Coal Co. has installed 1300 sets of these arches. The average installation cost per set is shown in Table 2.

Illustrations of the arches are shown in Figs. 10 and 11. They have been installed at varying depths below the surface to a maximum depth of 890 ft. While heavy pressure has slightly deformed some of the sets, practically none has had to be replaced.

10

11



FIGS. 10 AND 11.—STEEL ARCH-SUPPORT SETS, BENNETT BED, LOREE NO. 2 COLLIERY.
Fig. 10, shaft level east gangway.
Fig. 11, shaft level west gangway.

TOUSSAINT-HEINTZMANN STEEL ARCH SETS

Recently a new type of steel arch for roof support, known as the Toussaint-Heintzmann system, used extensively in Europe, was installed in No. 11 slope, heading No. 32, Clark bed, Marvine colliery, in North Scranton.

Each of these arches consists of three individual U-shaped sections; viz., the top or arch piece and two curved legs. The fundamental principle of the U-shaped section is to provide a stable arch resisting

lateral bending, in order to avoid lateral distortion and the resultant weakening of the arch. The U-shaped arch piece and leg sections are assembled in such a manner that the segments, separated by small wooden blocks, fit into each other and are held together by heavy clamps or U-bolts. These connections are known as "yielding" or "friction" joints (Fig. 12). The joints have very high frictional resistance and are calculated to give a sliding movement only at the moment when the various parts of the arch are stressed to their full limit and before defor-

FIG. 12.

FIG. 13.

FIG. 12.—SECTION OF TOUSSAINT-HEINTZMANN STEEL ARCH SETS AT MARVINE, SHOWING ALIGNMENT OF FRICTION JOINTS.

FIG. 13.—TOUSSAINT-HEINTZMANN STEEL ARCH SETS, NO. 11 SLOPE, HEADING NO. 32, NO. 1 COUNTER, CLARK BED, MARVINE COLLIERY.

mation occurs. These sets are designed to change only their circumference, under heavy pressure—not their shape.

The location at which these sets were installed (Fig. 13) is in a bed that originally was about 10 ft. thick, now crushed to about 5 ft. in height by heavy pressure brought about by failure of small irregular pillars. From November 1938 to May 1939 there were installed 100 sets, on 3-ft. centers, with heavy wood lagging and spragging, successfully controlling the surrounding rock packing.

An actual record of cost installation of the 100 sets discloses a total cost of $2544, or $25.44 per set, including labor and material.

Steel Tunnel-liner Plates

The steel liner plates were first used in 1927, in the Pioneer bore of the Moffatt tunnel in Colorado, which was driven through very soft ground that exerted full pressure all around the periphery of the opening. The maximum pressure observed reached 5½ tons per sq. ft. of surface, causing a failure of 12 by 18-in. fir timbers. This necessitated a lining that could withstand the terrific pressure continuously. The liner plates produced to meet this condition consisted of No. 12 gauge, 16 by 36-in. corrugated metal plates with flanged edges, bent to the radius of curvature desired. The flanged edges are drilled to permit bolting together to form a complete ring. In real wet conditions a monolithic installation was made by the addition of gaskets and additional bolts to give proper degree of tightness. Upon installation of steel lining, the voids behind were grouted with concrete and the inside was gunited.

One of the first installations of tunnel-liner plates in the Anthracite Region was at Gravity Slope colliery, at Archbald, Pa., in 1932, where an air bridge or overcast was constructed of such plates with bricked end walls (Fig. 14). The liner plates were installed on wood sills, and the entire installation remains intact to date. The total cost of the liner-plate installation, 18 ft. 8 in. long, 11 ft. 0 in. in diameter, was $285.60, or $15.30 per lin. ft. installed. The total cost of the entire overcast, including brick end walls, rock work for height, etc., amounted to $824.56.

During 1931 and 1932 an experiment was made by installing various types of roof-support material in a gangway at No. 4 slope, No. 4 east, Bennett bed, Boston colliery, at Larksville, Pa., which at that time was being reopened through caved ground, to determine the best methods of support and backfilling in broken ground. Four types of support were installed, alternating in eight sections, as follows:

	Feet
Tunnel-liner plates, 10-ft. dia	58
Schaefer lining, 10-ft. dia	125
Tunnel-liner plates, 10-ft. dia	38
Double timber, planked	75
Tunnel-liner plate, 10-ft. dia	67
Double timber, lagged	50
Tunnel-liner plate, 10-ft. dia	50
Tunnel-liner plate, 6-ft. dia	50
	513

The voids above and around this lining and timbering were backfilled with breaker refuse, blown into place by pipe by a low-pressure air current.

The colliery has been idle since October 1932, and the latest inspection, as of Aug. 22, 1938, discloses that after six years service the Schaefer lining is in excellent condition. The tunnel-liner plates have rusted somewhat, and in several places the wooden sills are beginning to rot. Lack of ventilation is the probable cause of the decay of the sills. The

section of double timber lagged has been crushed under a heavy fall, so that it is impossible to ascertain the condition of the liner plates beyond. The planked double timber has completely collapsed, owing to rotting of planks, which allowed fill to run through and clog the openings. It clearly demonstrates, however, that the steel and Schaefer lining are superior to wood for longer-life installation.

FIG. 14.—AIR BRIDGE OR OVERCAST OF STEEL LINER-PLATE CONSTRUCTION, NO. 5 SLOPE, ARCHBALD BED, GRAVITY SLOPE COLLIERY.

Table 3 gives a comparison of installation costs of the various types of roof support used in this experiment, the installations having been made by the colliery timbering forces:

TABLE 3.—*Installation Costs*

Roof Support	Man-hours at $0.6609	Cost per Linear Yard		
		Labor	Material	Total
Schaefer lining	44½	$29.42	$41.84	$71.26
Steel liner plates	31½	20.81	31.43	52.24
Double timber, planked	15⅓	10.13	10.53	20.66

At Baltimore colliery a further experiment was made with the liner plates, securing a manway through the surface wash. In this case the footings consisted of steel I-beams embedded in concrete header walls, to which the steel liner plates were bolted. This installation is 60 ft. long, 10 ft. in diameter, and cost $1,173.27, or $19.55 per lin. ft., installed.

To date, each of these installations of tunnel-liner plate is in good condition, showing no signs of failure. The plates were cleaned and painted with one coat, at the factory. A second coat of special paint, recommended by the manufacturer, was applied at the colliery before installation.

Precast Concrete Support Sets

On May 16, 1933, a U. S. Patent, No. 1909706, was granted to Paul M. Muspratt, of Kingston, Pa., for concrete mine-tunnel support structures, which are of superior strength, not subject to deterioration through exposure to mine air or water, and consequently form a more permanent installation.

The sets consist of a structure of precast and cured reinforced-concrete legs and collars, which are designed and erected to make the structure flexible and adjustable, in order that it may provide for the necessary

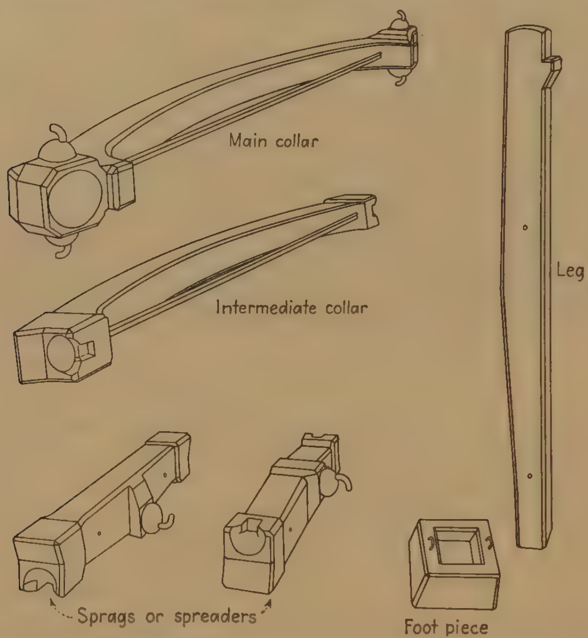


Fig. 15.—Individual members of precast concrete sets, Muspratt system.

yielding and distribution of strain in the event of severe localized vertical or side pressure. Briefly, a complete set of this type of support consists of three collars, two longitudinal sprags or spreaders and four legs. Two types of collars are available—the arched and truss-beam types. The collars are placed on 3-ft. centers while the legs are placed on 6-ft. centers.

The concrete members are reinforced with channels and rods and the forms are vibrated in pouring, in order to make the concrete as dense as possible and thereby reduce absorption to a minimum.

Fig. 15 illustrates the arch type of main and intermediate collars, leg, sprags or spreaders and foot piece. The main arched type of beam on the supporting ends is provided with concave seats in which the convex tops of the legs are placed to facilitate flexibility and distribute the load from

the collar to the leg, or any of the other joints, evenly through the total bearing area. Both vertical faces of the ends of the collar are provided with hemispherical projections equipped with stout steel hooks. The longitudinal sprags or spreaders contain reinforced socket ends into which the collars fit, thus making the unions a hooked ball and socket joint. The intermediate arched collar, being shorter, is similarly hooked to the spreaders, thereby making the entire installation connected by flexible joints. The feet of the legs are placed in reinforced-concrete box receptacles, provided with a cup-shaped recess with sloping sides, having an open space between its sides and the sides of the leg. Wooden or Celotex compression blocks are placed in the cup and the legs are placed thereon. A flexible sealing compound is placed in the voids around the leg to pre-

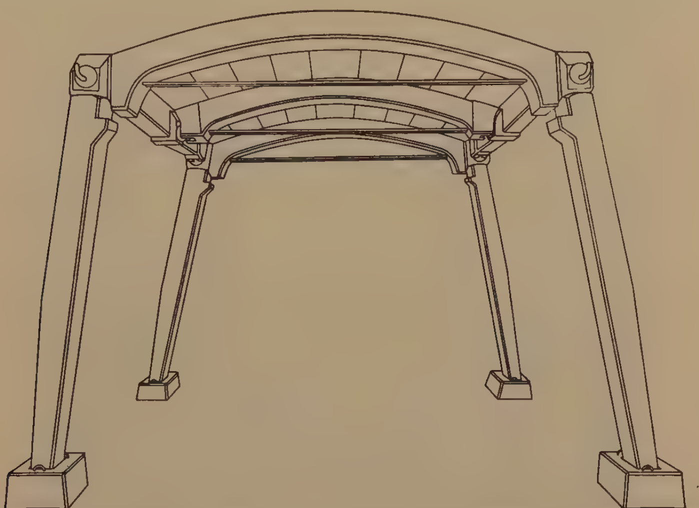


FIG. 16.—COMPLETE PRECAST CONCRETE SUPPORT SET, MUSPRATT SYSTEM.

vent water from affecting the feet of the legs. The top lagging merely comprises reinforced concrete slabs 12 by 36 by 3 inches.

When the timbering structure has been set in place (Fig. 16), wooden wedges are fitted and driven between the lugs on the collars and legs, thereby unifying the structure and providing for distribution of any strain that may be put on it by a side thrust of the ribs.

Table 4 is presented to show the comparative weights (approximate) of the concrete members and wood.

A test conducted at Fritz Laboratory, Lehigh University, shows that the ultimate concentrated load at the crown of the collar was 25,000 lb. applied on a top surface area of 1¾ sq. in. This member failed by cracking at that pressure. All the members of a support set, however, are manufactured of concrete having a compressive strength of about 6000 lb. per sq. inch.

TABLE 4.—*Comparative Weights of Supports*

Support	Weight, Lb.		
	Concrete	16-in. Round Oak, White	16-in. Round Fir
9-ft. collar	750	770	600
12-ft. collar	850	960	750
7-ft. legs for 9-ft. collar	300	448	350
7-ft. legs for 12-ft. collar	450	448	350

In 1932 the Glen Alden Coal Co. installed six complete sets of this type of timbering in the No. 2 tunnel, Bennett bed, Woodward colliery,

FIG. 17.—ARCH-BEAM COLLARS, PRECAST CONCRETE SUPPORT SETS, NO. 64 TUNNEL, WOODWARD COLLIERY.

near Wilkes-Barre. Upon completion, this portion of the mine became flooded, the sets being submerged under 90 ft. of water and remaining so for a period of 18 months. Upon dewatering, the area supported was found intact; the second set had been pushed slightly, proving its flexible properties. The surrounding territory, however, was badly caved, thus putting the installation to a very severe test.

The water in this territory is highly acidic. Upon dewatering there was about an inch of "yellow boy" covering portions of the sets but when

this was scraped off the concrete was found to be in a perfect state of preservation.

About 1000 ft. of the arch-beam type of precast concrete supports was installed in 1935 (Fig. 17) in No. 64 tunnel, Red Ash to Bennett bed, Woodward colliery, Wilkes-Barre, Pa. This also was flooded at the lower end with no apparent damage to the supports, as revealed when it was recently dewatered. The upper end, is being extended for an additional distance of 600 ft. with the truss-beam type of collar (Fig. 18). The reason for the change in the kind of collar is a low, broken, slabby roof, where it is more advantageous to use the flat collar because it requires less roof height.

FIG. 18.—NO. 64 TUNNEL, WOODWARD COLLIERY, TRUSS-BEAM COLLARS.

The cost of installing a concrete support set is approximately twice that of wood, but since the maximum life of timber is not over seven years, the use of the concrete construction is justified in locations having long life. In large quantities, naturally, the ultimate cost would be considerably less than where only a small installation is required.

It is to be understood that the general principle of the design is not, and cannot be, standardized, because of the variable conditions met. The general design therefore is flexible enough to meet those conditions as they arise, or to meet the standards of the mine in which they are to be used.

CONCLUSION

There may be other forms of improved roof support prevalent in the Anthracite Region, which the writers have not had an opportunity to investigate. It is, however, apropos to comment briefly upon collapsible steel props or jacks, which are in daily use in longwall operations and are indispensable. They were not included in this paper because they are classed as only temporary forms of roof support.

Acknowledgments

The authors take this opportunity to express their appreciation to Messrs. Paul Muspratt, inventor of the Muspratt system, W. C. Monroe, General Manager of the American-Schaefer Lining Co., H. H. Otto, Mining Engineer, and E. W. Stahl, Assistant Mining Engineer, of The Hudson Coal Co., for their valuable assistance and information in assembling the descriptions used in this paper.

DISCUSSION

(*C. A. Gibbons presiding*)

H. N. Eavenson,* Pittsburgh, Pa.—About 12 years ago the speaker visited a group of German mines in the Ruhr, where steel arches of various kinds similar to those shown had been largely used. Without having any accurate figures now, there were certainly several thousand yards of headings protected by this system. At shaft bottoms and permanent sidetracks the protection was usually by brick or block arches in which compression joints of creosoted timber, about 2 in. thick, were used both horizontally and vertically.

H. H. Otto,† Scranton, Pa.—Professor Bucky asked whether there has been any experimenting with the common three-piece timber set whereby some system of flexible legs could be used to permit slight movement of the overlying mass. I know of no experimentation of this nature, which, in all probability, might be due to the comparatively short life of timber. Since the February meeting, however, I have read an article in *Coal and Iron*, an English publication. The flexible prop consists of a prop set in a short piece of pipe of similar size, the bottom of the pipe being partly filled with sand, shale or some other compressible material, and the entire unit wedged in place with cap piece. The compressible material gives slightly under roof pressure of 20 tons. This prop, however, was designed for conservation of timber during the war, as the prop can be reclaimed by raising the pipe with a bar, permitting the compressible material to collapse and loosen the unit.

Some question was also raised regarding excessive pressures on flexible roof supports. In the Woodward mine of the Glen Alden Coal Co. there was a fire and squeeze, necessitating the flooding of part of the mine. There were various forms of roof support in this section—brick and concrete arches spaced to meet certain conditions, steel sets, Schaefer lining and Muspratt precast concrete sets, all of which were submerged for a period of about 18 months. Severe roof pressure followed dewatering, as the surrounding territory was caved tight. Both types of the flexible supports were in an area where the caved roof consisted of a laminated shaly structure; both remained intact and particular notation was made of the Schaefer lining, which cracked only slightly a short distance inside of both ends of the section.

C. Evans, Jr.,‡ Scranton, Pa.—The form of roof support described by Wirth and Dennen was adopted because of our seeing continual references to it in English technical papers. The first arches were installed directly on the mine floor. We soon found that the feet of the arches were very quickly affected by heavy corrosion,

* Eavenson, Alford and Auchmuty.
† The Hudson Coal Co.
‡ General Manager, The Hudson Coal Co.

but we prevented this in a simple manner through a scheme devised by one of our men, illustrated in the paper.

These arches invariably are installed at points where the roof is bad, and the illustrations in the paper show the method of lagging the vertical walls as far up the arch as is possible, and by slabs of rock supported on cross members of steel above these vertical walls, but there is no illustration of an even better method; viz., the lagging of the exterior of the arch with scrap sheet steel—in our case, shaker-chute pans, of which large quantities are available.

Questions have been asked as to the comparative value of these various forms of support. It is obvious that all of them are fairly expensive and can be justified only where there is a considerable life.

Our experience shows that the steel arch is the least expensive, with the steel liner plate next, concrete timber such as the Medusa precast and Muspratt next, and the Schaefer lining next, with the last three costing about the same per yard. We regard the Schaefer lining as the best, particularly at points where there is a strong probability of heavy pressure in the future. We have used all of these systems except the Muspratt, and we have seen failures in all the systems we used except the Schaefer lining.

P. Weir,* Chicago, Ill.—While confined to a discussion of the materials, steel and concrete, this paper really has to do with permanent and semipermanent roof support. The authors' comments on the use of wood have to do almost entirely with untreated wood, which, because of rapid deterioration, is not, except in rare instances, satisfactory for permanent and semipermanent support. However, timber that has been properly pressure-treated with a standard preservative is often a suitable and economical material and to eliminate it from consideration for use for permanent and semipermanent support is to overlook an opportunity for savings. I do not mean that treated timber has a universal application, neither do I believe that steel and concrete furnish a universal answer. Sometimes a combination is indicated. An example is the use of treated lagging behind steel sets. The comparison of steel and concrete with untreated wood is one thing; a comparison with treated wood is another. The authors' comparisons consider untreated wood only.

Going back some 12 years, I have been in contact with millions of feet of pressure-treated mine timbers used for roof support. These timbers have already outlasted several times the untreated wood timbers that formerly were used. I feel keenly, that any discussion on the roof-support problem that overlooks the durability of properly pressure-treated mine timbers is incomplete.

W. W. Wirth and W. L. Dennen (authors' reply).—We read with particular interest the comment by Mr. Paul Weir concerning the use of treated timber in the mines. We intentionally omitted treated timber from the paper because in long-life locations of mines where unusually heavy pressures are exerted or squeezes occur the experience in the Anthracite Region indicates that the treated timber does not resist pressure any better than untreated timber. We do find, however, that the treatment is a preservative and, therefore, in the natural course of events the treated timber outlives untreated timber for some years.

We have had actual experience in mines of The Hudson Coal Co. and elsewhere in the Anthracite Region where treated timbers and untreated timbers on the same haulage roads collapsed when pressure occurred in a particular area. Some of the treated timber varied in diameter from 12 to 20 in. There is no question that, where ordinary

* Mining Engineer.

roof pressures occur, treated timber has its place in the mines of the Anthracite Region as well as in the Bituminous Region.

The paper was prepared to present to the engineers methods of roof support where unusually heavy pressures or bad conditions prevail. There is no uniform answer as to what type of roof support should be used. Each location must be carefully studied and the type of roof support used must fit the local conditions and the economics of the area which, of course, takes into consideration the life of the area.

Economics of Wood Preservation in Underground Coal Mining

By Reamy Joyce,* Member A.I.M.E.

(New York Meeting, February 1939)

Data Required

Conditions in underground mining are so variable that in approaching the problem of the economies effected by the use of pressure-treated mine ties and mine timbers, it is necessary to secure specific data for each mine. Facts must first be dug out to show the service life of untreated timber and the reason for its failure. Generally speaking, decay is the usual cause of failure, but in some mines mechanical destruction is an important factor. Occasionally the two causes combine to produce failures, and it is difficult to determine positively which factor is responsible.

In making comparisons, it is always necessary to start with the cost in place and for this reason there must be accurate information covering the labor cost of installation and the labor cost of renewals.

When treated timber is used, it is necessary to know the specifications for the treating process, whether the timber was painted, dipped, treated open tank, or impregnated under pressure according to standard specifications. It is also necessary to know the preservative used and the amount of the preservative that was retained by the timber after treatment. It is also well to know whether the timber was thoroughly air-seasoned before treatment or was green at the time it was treated.

The kind of timber available, both untreated and treated, must be known and all of the factors listed above must be taken into consideration in the study of any problem.

Variable Conditions

A study was made recently of the untreated mine ties used by a bituminous coal company in two of its mines, which are about 20 miles apart. The mine ties were secured from the same sources for both mines. A careful check was made over a 10-yr. period on the life of untreated main-haulage ties, and it was found that in the mine where the haulage roads were on intake air the average life was approximately 5 yr., while in the other mine, where the haulage roads were on return air, the average life was 2 yr. A similar variability in untreated timber

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life has been noted in different parts of the same mine. A mine study recently showed an average life of 3 yr. for the best white oak mine timber installed in a drainage heading on return air, which compares with some white oak timbers on a main haulage road close to intake air, which had been in service for 18 yr. without evidence of failure.

If untreated timber is failing from decay, it is now generally recognized that standard pressure treatments with preservatives such as creosote, zinc chloride and Wolman salts will be effective in preventing decay underground. With these preliminary factors in mind, we will now consider the economies of pressure-treated timber installations underground and present the findings from a number of different angles.

Main-haulage Ties Treated with Zinc Chloride

During three years (1928, 1929 and 1930), 223,258 hardwood mine ties impregnated by the full cell pressure process with a net retention ½ lb. zinc chloride per cubic foot were installed on main haulage in a group of bituminous coal mines. These treated ties cost $0.7133 delivered at the mines. They averaged 15.45 feet board measure per tie. The cost of the untreated ties was $0.4017 or $26 per 1000 f.b.m. The cost of the treatment and delivery averaged $0.3116 per tie, which was $20.17 per 1000 f.b.m. The additional cost of these 223,258 pressure-treated mine ties was $69,602.53.

A modern pressure-treating plant had been built near these mines, so that the freight cost of making delivery was low. The additional cost of the treated ties compared with the untreated ties being used currently was 77 per cent. The actual additional cost of the treated ties in track, which is the correct starting point to use, was approximately 34 per cent.

It is not possible to identify these ties in track today. Most of the ties were installed in ordinary renewals, which means that they were scattered throughout the mines. It was only in advancing, where new track was laid, that it would have been easy to keep records. There is no evidence that any of these ties have been renewed. The physical examination of several hundred ties made during October 1938 indicated that they were in good condition and should have an added life at least equal to that of untreated mine ties, which had been determined at 3.7 years.

These treated ties have accumulated an average life to date of 8.78 yr., so that it is evident that an average life will be obtained of at least 12 yr. and it is probable from what we know of service records of material treated with zinc chloride in surface tracks that the average life will be greater than this.

Two renewals of these 223,258 ties have already been saved. Figuring that these ties, if used untreated, would have been replaced with the same

size and kind of ties that were installed treated, but at a cost of $19 per 1000 f.b.m. instead of the 1928–1930 cost of $26, we would then have the following renewal cost per tie:

Cost untreated tie	$0.29355	
Labor renewal	0.488	
Spikes	0.04	
Cost in track	$0.82155	
First renewal 223,258 ties @ $0.82155		$183,417.61
Second renewal 223,258 ties @ $0.82155		183,417.61
Saving to date, based on average 12-yr. life		$366,835.22

This saving represents a profit to date on the added cost of the treated mine ties installed during the years 1928 to 1930 of 527 per cent.

It is interesting to note from a study of these figures that the added cost of the treated tie is more than offset by the saving of the cost of the first renewal of an untreated tie.

In the four years, 1931 to 1934 inclusive, 95,689 treated main-haulage ties were installed. These ties have saved one renewal, so that the added cost of these ties has already been returned with a profit.

The treated main-haulage ties used in the 3 yr. 1935 to 1938 are still out of pocket as far as the additional cost is concerned, but are in process of producing the same savings.

Since 1928 these mines have operated approximately 70 per cent of capacity. The stand-by value of the treated mine ties during the equivalent of 3 yr. idle time is important to remember. If untreated ties had been used, decay would have progressed during the idle 3-yr. period to the point where their value as ties would have been almost completely gone. The treated ties, being free from decay, did not suffer during the idle period.

This straight-line method of figuring is simple and presents the savings without taking into consideration the fact that money has an interest value. In figuring the annual costs of using various materials such as treated and untreated timber, these interest costs must be recognized.

Annual Charge Method

The annual charge formula is expressed by the equation

$$A = \frac{P(1 + r)^{nr}}{(1 + r)^n - 1}$$

in which:

A = annual charge,

P = amount of initial investment,

n = number of years in the recurring period (the average life of the timber),

r = the rate of interest expressed decimally.

This method assumes that the money is borrowed to make the original installation and that the annual charge consists of the sum of simple interest on the amount borrowed plus a sinking fund calculated at the interest rate compounded semiannually, which at the end of the useful life of the installation will repay the amount borrowed.

With interest at 4 per cent, by the use of this formula, the annual charge per dollar of cost in place is as follows: 4 years, 0.27549; 12 years, 0.1065522.

In 1928 if calculations had been made based on the annual charge formula and a projected life of 12 yr. for the treated ties and 4 yr. for the untreated ties, the annual charge per tie would have been:

	Cost in Track	Life, Yr.	Annual Charge
Untreated tie	$0.9297	4	$0.2561
Treated tie	1.2413	12	0.1323
Annual saving per tie			$0.1238
Saving per tie, 12 yr			$1.4856
Saving 223,258 ties, 12 yr			$331,672.08

It is probable that the actual average life of these treated ties will be nearer 16 yr. than 12 yr. and that the saving therefore will be greater.

Main-haulage Ties, Creosoted

In December 1917, on the main bottom in a new bituminous coal mine 3809 sawed southern pine mine ties, 5 by 7 by 6 ft., impregnated by the Rueping pressure process with 8 lb. creosote per cubic foot of timber, were installed. These treated ties cost $1.13 each delivered at the mine. The cost of the untreated mixed hardwood ties that were available would have been $0.28 each. The creosoted ties were protected by tie plates ½ by 6 by 8 in. These creosoted ties with the tie plates amounted to 508 per cent of the untreated ties. The cost of these creosoted ties, tie-plated and installed in the track, was 178 per cent of untreated ties. The estimated life of the mine was 20 yr. The labor cost of installing ties in new track was estimated at $0.50 per tie. The labor cost of renewing ties was estimated at $0.80 per tie.

The use of creosoted ties was justified by the following calculations. The expected life of untreated hardwood ties was estimated at not over 4 yr. The estimated life of the creosoted pine was not less than 20 yr.

Untreated hardwood, cost in track	$0.78
Four renewals @ $1.08 each	4.32
Total 20-yr. cost for untreated ties	$5.10

Creosoted southern yellow pine, 5 by 7 by 6 ft	$1.13
Two ½ by 6 by 8-in. tie plates @ $0.147 each	0.294
Labor installation	0.50
Total	$1.924
Saving	$3.176

This simple straight-line method of figuring does not include interest. When these same data are used with the annual charge equation and interest at 4 per cent, the following figures are obtained:

	Cost in Track	Annual Charge
Untreated hardwood	$1.08	$0.297
Creosoted pine	1.924	0.14237
Saving per tie per year		0.15463
Saving per tie 20 yr		3.0926
Saving 3809 ties 20 yr		$11,779.71

Twenty-one years have now passed since these ties were installed. One hundred creosoted ties had to be renewed in 1935 after 18 yr. of service. Derailments at a crossover had widened the track gauge so that decay reached the untreated interior of the ties through the spaces around the spikes. An examination made in December 1938 found 10 additional rotten ties in track.

During these 21 yr. there has been the equivalent of 5 yr. of idle time in the operation of this mine, in which time these creosoted ties have continued to resist decay, again indicating the high stand-by value of pressure-treated wood. Last year additional coal lands were secured, so that now the estimated life of this mine is another 15 years.

From the inspection just made, most of these creosoted ties look as though they would give many years of additional service.

Wooden Bottoms, Steel Mine Cars

Many bituminous mines are using composite mine cars, which consist of steel bodies with wooden bottoms and bumpers. The steel bodies are subject to corrosion. In wet mines where the coal is high in sulphur, corrosion proceeds rapidly. Coal-mine operators and mine-car manufacturers that have studied the question of corrosion have estimated the life of steel bodies in the following order: plain steel, 10; copper-bearing steel, 12; alloy steel, 16. It must be remembered that where plain steel bodies have failed from corrosion in 10 yr. in mines where conditions are unfavorable, cars of the same type have lasted 50 to 100 per cent longer in dry mines. Formerly most mine managements believed that untreated wood mine-car bottoms failed from mechanical wear. Now a number of coal companies know that when preframed creosoted oak bottoms are used, mechanical destruction, so-called, does not take place

at the time when the untreated bottoms failed, but is postponed for a considerable period.

One of the large bituminous coal companies placed in service 1500 composite mine cars in 1929, of which 300 had preframed pressure-creosoted pine bottoms and the remainder had pressure-creosoted oak bottoms. In the 9 yr. that these cars have been in service, the pine bottoms are showing definite signs of failure from abrasion. The oak bottoms still show very little evidence of wear. The actual average life of the creosoted oak bottoms is not yet known. It will probably be not less than 12 years.

The approximate cost of a 4-ton composite mine car with copper-bearing steel sides, steel wheels, antifriction bearings and untreated white oak bottom, is $275. The steel sides of the car would have an estimated life of 12 yr. White oak bottoms untreated have produced a life of from 5 to 6 yr. Mixed oak bottoms untreated last about 4 yr. Untreated oak bottoms fail primarily from decay. The added cost of preframed and bored creosoted oak bottoms in new cars is $8.50 and the life of this bottom is estimated to be the same as the life of the copper-bearing steel. This added cost is only 3 per cent of the cost of the composite car with the untreated wood bottom. The replacement costs of wood bottoms in cars in use are:

	Preframed Creosoted Oak	Untreated White Oak
Lumber	$11.50	$ 4.75
Bolts and nuts	3.75	3.75
Labor	9.68	11.06
Cost per car	$24.93	$19.56

The "capitalized cost" method of cost comparison is the first cost plus the sum of the present worth of all successive costs. This method has a particular application when the life of the installation is limited by a definite number of years. Using this method, we arrive at the following capitalized cost comparisons:

	Preframed Creosoted Oak (12-yr. Life)	Untreated White Oak (6-yr. Life)
Original added cost	$8.50	$ 0.00
Present worth @ 4 per cent		
1st renewal, 6 yr	0.00	15.46
12-yr. cost	$8.50	$15.46
Saving per car		$ 6.96

Since the company has 6000 composite mine cars in service, the total saving is $41,760. This amount is the present worth of the saving resulting from the use of pressure-creosoted mine-car bottoms.

Mine Timbers

In 1928 a representative bituminous coal company that was fortunate in having tributary to its operations a splendid quality of oak timber completed an 8500-ft. waterway heading. The following information is extracted from the engineering notes made on this project:

This heading was a drainageway for the water from four large coal mines and due to roof conditions was timbered with 1400 three-piece timber sets. The life of white oak timber based on experience in main headings was estimated at 10 years. The timber which was required for this heading had been carefully produced, stacked up and thoroughly air seasoned before being installed. In the third year after this timber had been in place, numerous failures from decay occurred and it was definitely determined that due to the moisture and humidity conditions and the fact that this heading was about two miles from the air intake the decay hazard was unusually severe.

Not much was known about pressure-treated timber, but it was felt that a life of six years could reasonably be obtained and, therefore, on this basis the use of pressure-treated three-piece timber sets was started. Excellent records were kept of the creosoted sets and of the sets treated with various salt preservatives. All of the treated timber was preframed before treatment.

The untreated oak sets cost including framing \$5.73, cost of labor for installing each set, \$1.75, making the cost in place \$7.48. The pressure-treated sets delivered at the mines in car-load quantities cost \$6.98. Their cost in place was \$8.73 or \$1.25 more per set than the untreated timber. By using the capitalized cost method, we have the following figures:

	Untreated Timber	Treated Timber
Initial cost per set in place	\$ 7.48	\$8.73
Present worth of renewal 3 yr. hence at 4 per cent	6.65	
Present worth of cost to assure 6 yr. protection	14.13	8.73

\$5.40 per set is the present worth of the saving effected by using treated timber if same has an average life of only six years.

Six years of service have been completed and the pressure-treated timber is still free from decay.

When these treated timber sets were first installed, no provision was made for treated cribbing, lagging or wedges to support the roof above the crossbars. In 1934, or 3 yr. later, the untreated cribbing began to fail from decay, and there were many falls, some of which knocked down the treated sets. This condition was recognized promptly and all of the cribbing, blocking, and wedges above the treated sets was replaced with treated material ordered especially for that purpose. Since that time there has been no trouble from this cause.

From what is known of the service records of creosoted material in surface use, it may be assumed that the timbers treated with creosote should be good for the life of the heading, which is estimated at 20 yr. In this event, using the annual charge formula, we arrive at the following projected savings:

	Cost in Place	Life, Yr.	Annual Charge
Untreated	$7.48	3	$2.695
Creosoted	8.73	20	0.642
Annual saving per set			$2.053
20-year saving per set			$41.06
20-year saving 1400 sets			$57,484.00

The life of the salt-treated sets is difficult to project for the reason that one leg of the three-piece set is in the water ditch and is, therefore, in contact with running water, and for this reason a failure from decay is anticipated in this leg earlier than would ordinarily occur.

Re-use Value

In 1926 one of the larger bituminous coal companies began the use of creosoted main-haulage ties and since that date has installed over 500,000 pieces in its mines. In 1936 about 600 creosoted ties were removed when the rail was taken out of a main-line siding that had been installed in 1926 and was no longer needed. They were examined and found to be in perfect condition. The spike holes were plugged with creosoted plugs and the ties were sent back into the mine for further use.

Another representative company, which has over 300,000 creosoted mine ties in use, also started the use of creosoted main-haulage ties in 1926. This year it recovered some 60-lb. rail from one of its old mines. After nine years of service, 485 creosoted main-haulage ties were brought out, examined and found to be in excellent condition. The spike holes were plugged with creosoted plugs and the ties are now again in track in another location.

In 1926 a bituminous coal company that was operating on a hand-loading basis began the use of 3 by 5-ft. working-section ties, treated with zinc chloride, for use in entries and rooms. About 60,000 of these ties were installed in two mines in 3 yr. When the operations were mechanized 10 yr. later, this practice was discontinued, but it was found that during the 10-yr. period many of these ties had been re-used four times.

In 1934 a group of three bituminous mines started the use of creosoted 3 by 5-ft. working-section ties, installing approximately 60,000 in that year. These ties are still in use, some now being in their seventh location. These operations are on a hand-loading basis.

In both of the last two instances, the companies are planning to use the treated ties as cribbing and blocking when they have become spike-killed, which indicates that they have an important salvage value.

Service Records

The examples of economies cited are made possible by the definite data available on the service records of both the treated and untreated material. Without accurate records to start with, such calculations would not be of any value.

A carefully thought out system of keeping service records of treated mine ties and timber installations in bituminous coal mines is important to the mines themselves and to the coal industry. The opportunity that many of the coal companies have to install test lots of treated mine ties and mine timber, all properly identified as to the complete treating record and the date of installation, would make valuable data that could be looked upon with confidence. Such a plan would also make possible regular check inspections of these test installations jointly by the operating and engineering departments, so that as time went on the causes of success and failure would be known. Such a program is earnestly recommended.

DISCUSSION

(*W. H. Lesser presiding*)

H. H. Otto,* Scranton, Pa.—About 10 or 12 years ago, while studying the question of treated ties for standard-gauge tracks, I walked sections of railroads operating in the Lackawanna and Wyoming Valleys and examined a large number of untreated ties that had date nails in them. I found a number that had been in use about 16 yr. and were still in reasonably good condition. Most of them were long-leaf yellow pine.

Based upon this study, creosoted ties were placed in sections of our standard-gauge track from a point just above the empty scales above the breaker and the loaded scales below the breaker in several of our long-life operations. In the empty and loaded car yards, where the traffic is not so heavy, it was felt not advisable to put in treated ties. However, some of the untreated ties being purchased today are inferior in durability, when compared with those purchased 25 yr. or more ago. It may be necessary to use treated ties throughout the empty and loaded yards where the life of the operation exceeds 15 years. Because of the ample supply of untreated ties at a low cost, it does not pay to install treated ties in the mines in this region.

In a number of instances, experience has shown that treated timber, while it does not rot, has broken more quickly by crushing than untreated timber in sound condition and containing some sap. On the other hand, there are locations where treated timber and untreated timber were stood shortly before a mine was shut down and because of the lack of ventilation the untreated timber soon decayed while the treated timber is still standing with apparently its original strength.

There seems to be no hard and fast rule for the use of treated ties and treated timber. Each problem must receive careful and separate study.

R. Joyce (author's reply).—I agree with Mr. Otto that each timber problem in connection with coal mining should receive careful and separate study.

Mr. Otto said that it does not pay to install treated ties in the mines in the region referred to. This statement should be modified in accordance with his statement regarding study. Often the life of the native timber is so short that even if it did not cost anything it would be too expensive to use compared with the use of pressure-treated timber because of the high labor cost and frequent renewals required. The naturally durable woods that formerly were available in considerable quantity in the United States are no longer available at reasonable prices. The areas directly tributary to most of the important mining regions have been cut over for mine timber for many years, and the remaining stand is not as a rule of desirable species if used untreated.

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* The Hudson Coal Co.

Longwalling on Timber in Alabama Coal Mines

By L. I. Cothern,* Member A.I.M.E.

(New York Meeting, February 1940)

The introduction of mechanized mining has created a demand for long working faces. It has also prompted mining men to contest the old theory that longwall methods can be used only where roof conditions are ideal and where pack-wall material is plentiful.

"Longwall" as used in this paper means "modified longwall," "semi-longwall," or any of the various "wall" or "long-face" methods of mining. Many mines are now using long conveyor faces that differ only in their methods of development, types of roof support and method of loading coal onto the conveyor. Some types of longwall operations are being carried on successfully under conditions heretofore considered impossible by men experienced in longwall mining.

With possibly one or two exceptions, long-face methods in Alabama are a direct result of the need for lower mining costs. The advantages over room-and-pillar systems were clearly expressed in an article by Messrs. Fies and Lacy of the DeBardeleben Coal Corporation[1] as follows: (1) increased tonnage per miner; (2) increased percentage of lump coal; (3) reduction in cost of development; (4) elimination of the cost of room yardage; (5) reduction in the cost of haulage, from ability to use larger cars; (6) reduction in the cost of machine cutting; (7) reduction in cost for tracks; (8) reduction in the cost of deadwork; (9) reduction in the cost of supervision.

An attempt on the part of the operator to gain these advantages by changing from room-and-pillar to longwall requires considerable courage, and credit should be given to those men who have done their bit to meet the competition of rival fuels. The initiatory problems to be solved are in reference to: (1) retreating or advancing system, (2) length of wall, (3) method of roof support, (4) method of entry development, (5) type of conveyor or loader, (6) hand vs. machine loading (onto conveyors), (7) hand vs. machine loading in entries, (8) control of roof pressure,

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[1] References are at the end of the paper.

(9) effect of roof slips, rolls, partings, etc., on the new system, (10) adaptability of local labor to the new system, (11) safety to men.

This paper submits a brief summary of the methods being used by Alabama operators. Five mines, two of which have been previously described in mining journals, are used as examples. The mines selected

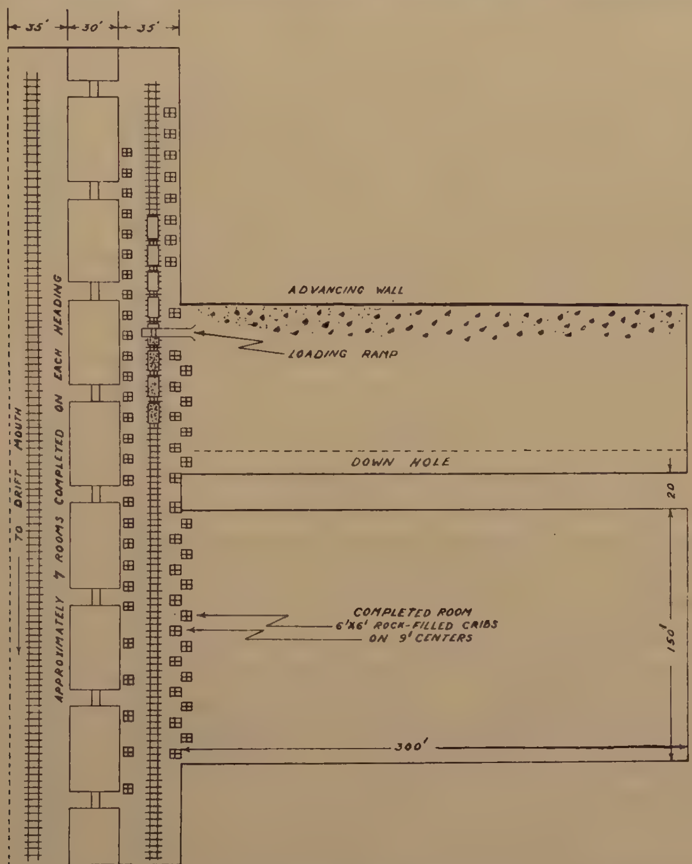

FIG. 1.—BARNEY MINE, ALABAMA BY-PRODUCTS CORPORATION. (FROM *Mechanization*, FEBRUARY 1938.)

are not the only ones using long-face methods, but are fairly representative of the methods used in Alabama. These methods may be classified as follows: (1) wide rooms with faces advancing, (2) wide rooms with ribs advancing, (3) retreating longwall, (4) advancing longwall. All mines described have working faces of 150 ft. or more in length.

BARNEY MINE

The Barney mine, in the eastern part of Walker County, operated by the Alabama By-Products Corporation, falls under classification 2,

"mining by wide rooms with advancing ribs." A complete description of the mine appeared in *Mechanization*,[2] so reference will be made only to physical conditions, method of mining and method of timbering. The upper bench of the Mary Lee seam (33 in. thick) is being mined under a cover ranging from 100 to 300 ft. Above the coal is a stratum of slate ranging in thickness from a few inches to 5 ft. and from 60 to 80 ft. of sandstone. Fig. 1 shows the room layout. "Down holes" are driven 15 ft. wide and to a distance of 300 ft. Production is started on the rib of the down hole, thus giving a working face 300 ft. long. The face is advanced until the roof shows evidence of breaking. The average advance is 150 ft. Another downhole is then started, with a pillar 20 ft. wide between it and the old face.

The entries are driven 35 ft. wide, and are held by a double row of cribs (9-ft. centers) on the room side and a single row on the pillar side. Enough bottom is lifted for a width of 7 ft. to provide a height of 6 ft. Split-timber props on from 4 to 6-ft. centers are used at the working face and left in place. Scrapers are used for loading. This method of mining has been used for several years, and apparently is satisfactory under the conditions for which it was adopted.

Praco Mine

The Praco mine, also operated by the Alabama By-Products Corporation, is in the Warrior coal field, Jefferson County, 38 miles west of Birmingham, Ala. The seam mined is the Mary Lee, which contains a top bench of coal averaging 48 in. in thickness, a hard shale parting of 22 in. and a lower bench of coal—a total thickness of 84 in. Only the upper bench is mined. The overburden, about 300 ft., consists of alternations of sandstones, shales, clay and limestone, with gritty sandstone at the base. The immediate roof is a hard shale, which has a good tensile strength. The bottom is the hard shale parting, which provides a strong floor.

The usual problems pertaining to longwall roof control in mines previously operated by room-and-pillar methods were encountered when it was decided to adopt wall-and-conveyor mining as a means of reducing costs at this mine.

The mine is opened by a slope pitching from 8° to 12°. Longwall work was started on the 13th and 14th south headings about one mile in from the slope mouth, all previous work having been done by room-and-pillar methods. Fig. 2 shows the plan used in developing the wall headings for retreating longwall work on the south side. The area near the high-water contour was mined by room-and-pillar on the 14th and 15th south headings as assurance against possible flooding. For the same reason only short walls were worked on the 13th south entry. Falls occurred after the 13th south wall had retreated a distance of 250 ft.

and the 14th south wall a distance of 153 ft. At the time of the fall on the 14th south wall there were 39,000 sq. ft. of roof, supported by split timbers, which gave no evidence of breaking until about 30 min. before the fall. Cribs had been placed along the entry, in accordance with the

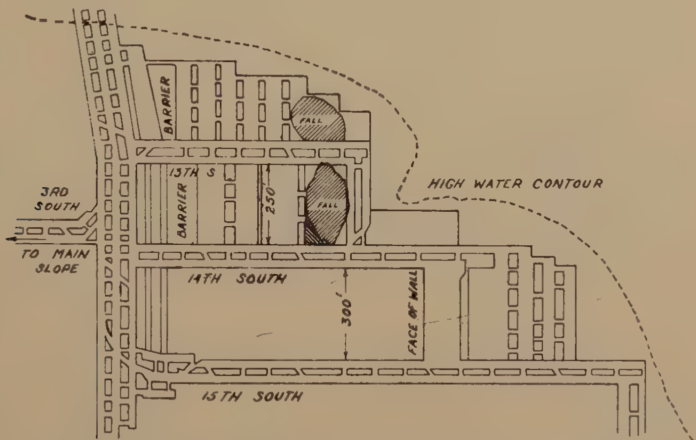

FIG. 2.—WALL DEVELOPMENT, SOUTH SIDE OF PRACO MINE.

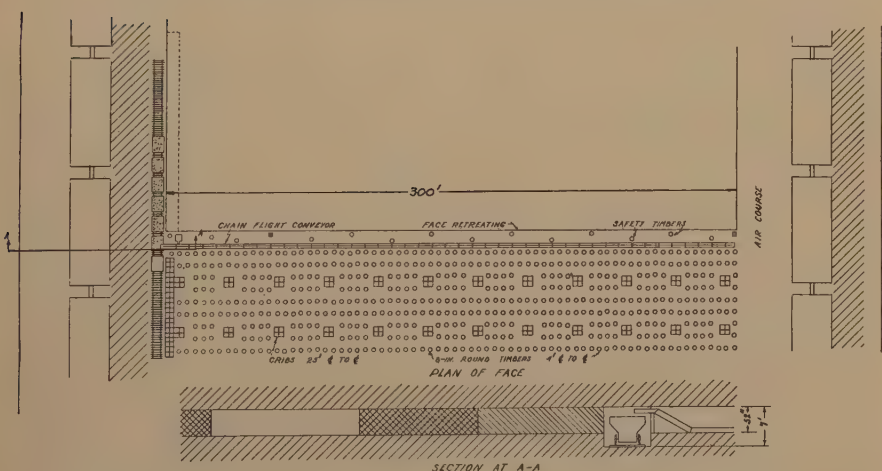

FIG. 3.—PRACO MINE, ALABAMA BY-PRODUCTS CORPORATION.

method used at the Barney mine. The fall rode the timbers toward the face of the wall and caught the conveyor. Part of this was recovered by breakthroughs driven from the new wall. The triangular fall shown occurred one day later, and the fall in the 13th south three days later. As a further experiment in long-face mining in this mine, the remainder of the 13th south was worked by driving rooms 50 ft. wide and using face conveyors and chain-flight room conveyors. This area is still

standing. The remainder of the 14th south was mined by the longwall method, the only changes being in the use of 6-in. round timbers and the leaving of small pillars after the wall had advanced approximately 100 feet.

Continuous retreating longwall was attempted in the 15th south heading. Fig. 3 shows the plan of the face and method of timbering. In the entry, bottom is lifted for a width of 7 ft., to provide headroom. The double entries are driven 30 ft. wide on 50-ft. centers, the coal being loaded by duckbill shaker loaders. The bottom rock is gobbed. A slab cut is made ahead of the wall to provide extra space at the end of the wall. The face is 300 ft. long and is undercut by a Goodman longwall machine having a 5-ft. 8-in. cutter bar. A Jeffrey chain-flight conveyor, No. 61AM, having special lap pans (not bolted), is used and is loaded by hand. Gathering locomotives pull the loaded cars to the main-slope sidetrack, from which they are hoisted to the tipple. Round timbers, ranging in diameter from 6 to 14 in., spaced 4 ft. along the wall and 6 ft. normal to the wall, supplemented by cribs on 25-ft. centers, are used to support the roof. The wall crew consists of one wall boss, one operator and a helper for the ore-cutting machine, one shot-firer and driller, one timberman, eight loaders and one entry man. Extraction is carried on continuously during two 7-hr. shifts.

At the time of the writer's visit, the wall had advanced 100 ft. with no visible evidence of roof weight. Later it advanced to a distance of 220 ft., still with no break, thus leaving a hanging roof of 66,000 sq. ft. The management then decided to force a break by shooting timbers out from the back side after first placing three rows of breaker timbers and one row of cribs to protect the face. After the removal of 75 ft. of timbers, the roof broke back of the breaker line. The face was started again and advanced 35 ft., at which time trouble was experienced with roof draw rock, some of the pans being caught by 3 ft. of this material. A 15-ft. pillar was then left and another wall was started. It is now the intention of the management to advance the wall about 165 ft. and leave another small pillar. The experience gained by this company with a strong roof that will stand over large areas and then break suddenly along the face should be of much value to others who are planning a mechanization program. Further experiments will be made to determine the best method of controlling this type of roof.

Mines at Boothton

The Southern Coal and Coke Co. operates three mines at Boothton, Shelby County, Alabama, and can be considered a pioneer of longwall mining in the state. The company's properties lie in the Cahaba coal basin, about 38 miles south of Birmingham. The eastern edge of this basin is sharply upturned and, for the most part, cut off by a fault. The

southern portion is irregular, having transverse faults and fault zones. The Gholson and Clarke seams, which are mined here, vary greatly in thickness in accordance with the characteristics of this basin. The longwall system was adopted here because room-and-pillar methods were practically impossible, owing to the fact that often pinching out of the coal made necessary the stopping of work. The miners would start rooms, advance them until the coal thinned out, then throw down their shovels until a better working place was provided. The company officials, having had previous longwall experience, decided that a longwall system would provide the necessary space for handling rock, place the miners on a daily wage, eliminate much narrow work (including some entry driving) and consequently develop a more satisfactory operation and a happier relationship with the miners.

Mine No. 1, *Boothton*

No. 1 mine was developed in the Gholson seam, which has a thickness of 32 in. at the present working faces. It has a shale floor and slate roof. The overburden is about 600 feet.

Fig. 4 shows the general plan of development. The mine opening is a slope driven directly down the pitch, which varies from 15° at the top to approximately 7° at the bottom. Wall entries are turned on whatever angle will give a slight grade in favor of the loaded cars. A pillar varying from 170 to 250 ft. is left to protect the slope. An up shot is then driven about 30 ft. wide from the lower wall entry to the upper. As this advances, the chain-flight conveyor is extended and used for loading out the coal. The length of wall varies slightly, depending upon the working conditions, the average length being about 300 ft. An attempt is made to complete an entire working cycle in a single shift. To eliminate entry yardage, the working face is extended across the entry and the top is taken down later for a width of 10 ft. The top rock is gobbed in the form of pack walls, as a means of keeping the entry open. The wall is then advanced until the property line is reached.

Fig. 5 shows the face details, conveyor layout and method of timbering. The face is undercut from *A* to *B* by a Jeffrey longwall cutting machine having a 5-ft. cutter bar. The timbermen follow the cutter. The cycle is then to shoot, load by hand onto the face conveyor, and move the conveyor forward. The wall crew for the day shift consists of wall boss, car trimmer, hoistman, one timberman and helper, gobber, machine man and helper, boom man and seven to nine loaders. The crew for the second shift consists of a shift leader and from three to five laborers. This crew moves the conveyor forward, takes down rock and handles other miscellaneous work that might interfere with the normal cycle of the wall crew.

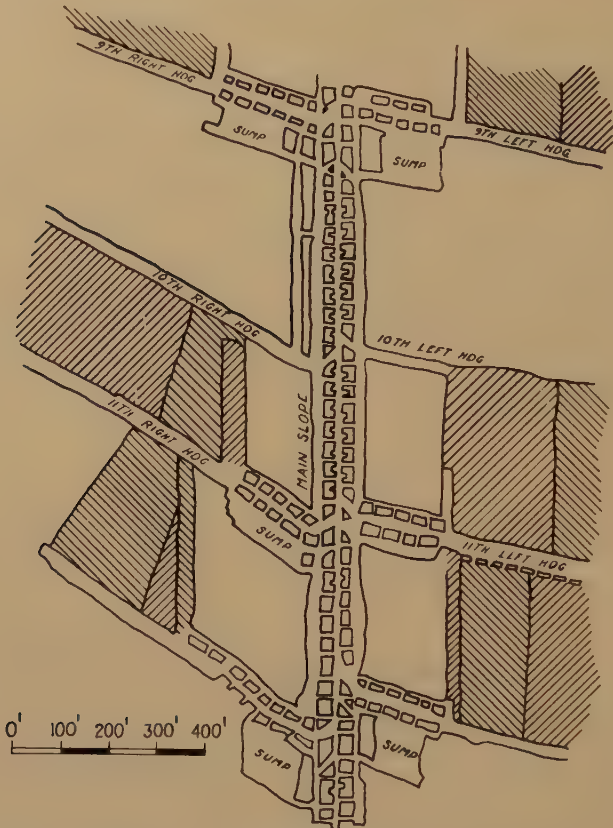

FIG. 4.—MINE No. 1, SOUTHERN COAL AND COKE COMPANY.

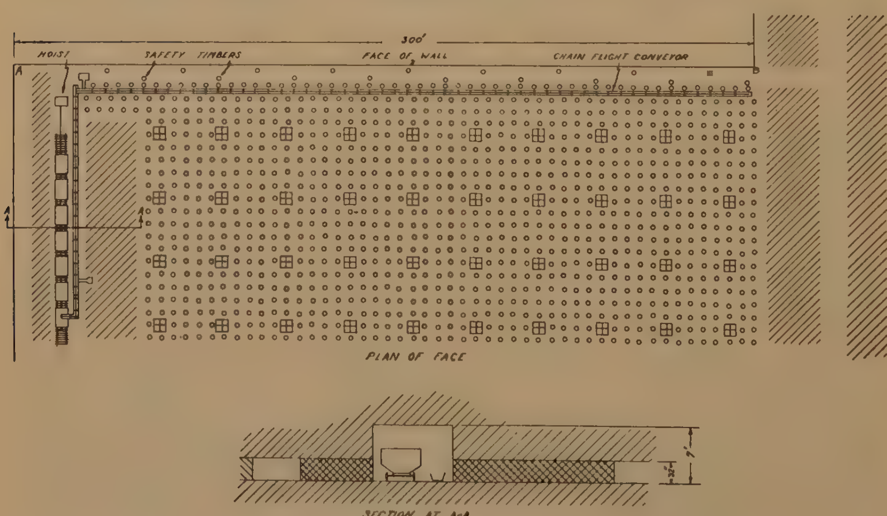

FIG. 5.—MINE No. 1, SOUTHERN COAL AND COKE COMPANY.

The conveyors and drives used in this mine were designed by the company. The pans are made from steel plate of No. 11 gauge, 36 in. wide and bent to give 12-in. bottom. The ends are lapped and not bolted. A 7½-in. Jeffrey chain (No. 104) is used. The face conveyor empties into an entry conveyor, which is placed along the gob line, and in turn it loads into the cars by means of a boom and cross table. A hoist on the track between the cross table and face of the wall lowers the loaded cars downgrade to a side track just off the main slope, and pulls the empties back upgrade into loading position. A main hoist on the surface hoists the cars up the main slope to the tipple.

The roof is supported by 8-in. round timbers placed on 4-ft. centers. In addition to these timbers, cribs are used when needed, and spaced to meet conditions of the roof action. The usual crib spacing is 25 to 30 ft. along the wall and 10 ft. in the direction of the advancing wall. They are placed closer together when the roof shows evidence of a possible break near the wall.

When pinched areas are encountered, the procedure is to brush enough roof to enable the cutting machine to cut the full length of the wall. An occasional exception to this is made when the coal pinches at the upper end of the wall with no recoverable coal above. In such case, the wall is cut short and the roof at the upper end of the wall is brushed for return air. In this manner the short wall is advanced until the pinched area has been passed.

Mine No. 4, *Boothton*

The Boothton No. 4 mine is opened by a slope driven in the Clarke seam, which lies below the Gholson. At the wall visited by the writer, the thickness of coal was 62 in. over all, including a 16-in. parting. The roof in most places is hard sandstone, with about 650 ft. of overburden. Fig. 6 shows the general layout of the mine. Under the coal seam there is a stratum of soft shale from 12 to 18 in. thick, which rests on 10 to 15 in. of coal. This bottom is taken up to provide headroom in the haulageways.

In this mine the headings are driven far enough in advance of the wall to give room for a trip of cars between the face of the heading and the conveyor loading boom. Fig. 7 shows an advancing wall and heading. The hoist is placed along the side of the heading, and to do this requires a head sheave at the face. The roof conditions require that the cribs be spaced closely in the direction of the advancing face. Fig. 7 shows that the inbye face of each row of cribs is roughly in line with the outbye face of the row ahead.

The system of haulage, cycle of operation and number of men per crew are the same as at No. 1 mine.

Empire Mine

The method of mining in the Empire mine, operated by the DeBardeleben Coal Corporation, is included in this paper to represent the use of "wide rooms with faces advancing." Complete details other than timbering are given in *Mechanization.*[2] The Black Creek seam, 32 in.

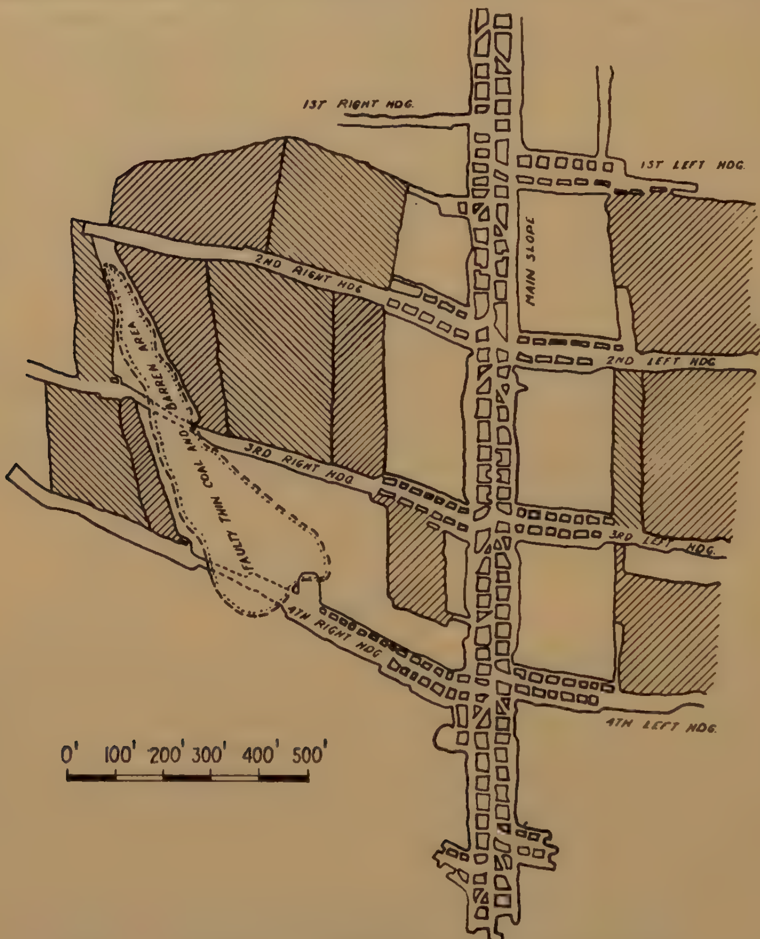

Fig. 6.—Mine No. 4, Southern Coal and Coke Company.

thick and having an overburden ranging from 40 to 200 ft., is mined by advancing rooms 150 ft. wide off panel headings that are driven on 550-ft. centers. Fig. 8 shows the room layout and cribbing. The coal is loaded by hand onto two face conveyors, which in turn empty into a main conveyor laid along the center line of the room.

The top is a sandy shale, which requires close timbering. Small pillars are left along the entry (sometimes supplemented by cribs).

Cribs are placed skin to skin along the main conveyor, and in rows parallel to the face on 100-ft. centers. Between cribs, posting is done to suit conditions.

Conclusion

The timbering methods just described indicate that in wall operations close posting is done with enough cribs added to suit conditions. In the wide rooms the amount of cribbing is fixed and the posting varied to suit conditions.

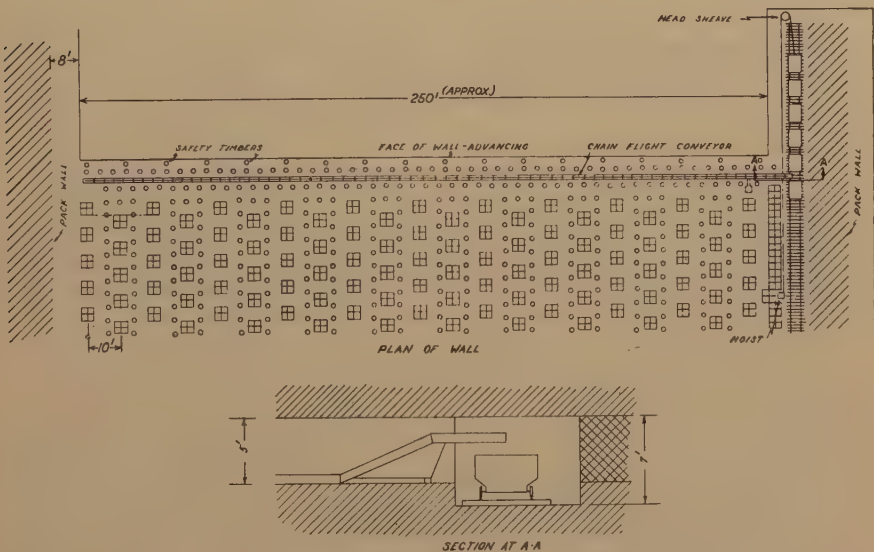

Fig. 7.—Mine No. 4, Southern Coal and Coke Company.

It should be kept in mind that the methods adopted were a result of much experimentation and were developed to meet local conditions. Studies of roof action for faces of various lengths and supported by both light and heavy timbers, with and without cribs, have been made by Alabama operators. Similar experiments should be made elsewhere before a final decision is made relative to a mechanization program.

The main problem connected with longwall in its early stages, especially when carried on by miners that have not had longwall experience, is that of protecting the working face. This problem was particularly complex at the Praco mine, where the roof gave more trouble after a break occurred than it did before. Until roof characteristics are known, a plentiful supply of timbers should be kept near the working wall to be used as emergency breaker timbers if the roof begins to work.

Though much timber is used in the mines described, the savings in the cost of cutting, haulage, explosives, labor and supervision more than offset this item of expense. Information relative to the reduction in

costs resulting from mechanized long-face mining is included in the article by Fies and Lacy.[1] Reference is also made in that article to the method of timbering used as protection against roof slips in the Sipsey mine of the DeBardeleben Coal Corporation.

Acknowledgments

The writer wishes to acknowledge his indebtedness to the mine officials named below, for their cooperation in furnishing the information, drawings and sketches needed for the preparation of this paper:

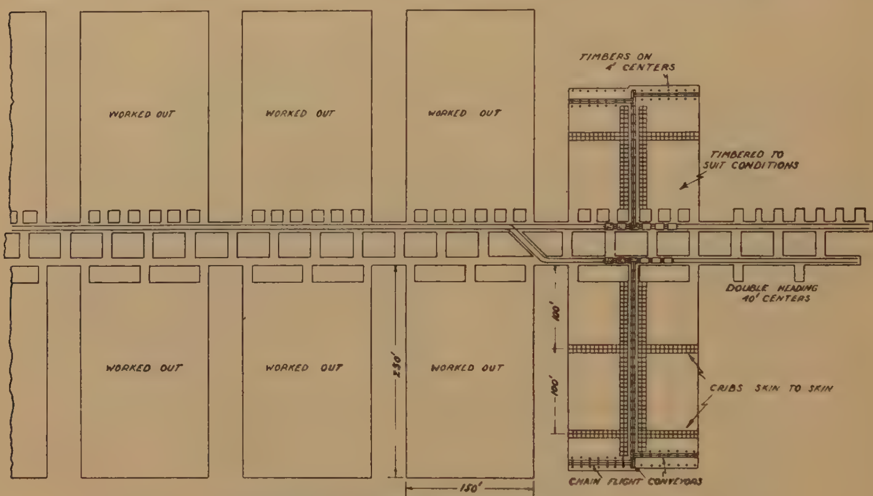

Fig. 8.—Empire Mine, DeBardeleben Coal Corporation.

Mr. P. H. Haskell, Jr., general manager of mines for the Alabama By-Products Corporation; Mr. W. C. Chase, general superintendent of that company; Mr. John Hager, superintendent of the Praco mine; Mr. George F. Peters, president of the Southern Coal and Coke Co.; Mr. Guy L. Chamberlain, vice-president, Mr. E. T. Hunter, secretary, and Mr. J. J. Beavers, superintendent of mines, Southern Coal and Coke Co.; Dr. Milton H. Fies, vice-president in charge of mining operations, DeBardeleben Coal Corporation. Mr. W. F. Diamond, senior mining engineering student at the Virginia Polytechnic Institute, made the final drawings.

References

1. M. H. Fies and W. M. Lacy: How Sipsey Mine Laid Plans for Mechanization and Mechanized with Profit. *Coal Age* (Sept. and Oct. 1931).
2. *Mechanization* (Feb. 1938) **2,** No. 2.

Effects of Artificial Support in Longwall Mining as Determined by Barodynamic* Experiment

By P. B. Bucky,† Member A.I.M.E., and R. V. Taborelli‡

(New York Meeting, February 1939)

This investigation was carried on by means of models and the application of the principles of similitude to determine the effects of props, props and cribs and sand filling in longwall mining. The geologic structure and rate of face retreat were kept constant and the only change introduced was in the method of support. It is not felt necessary to repeat a description of apparatus, or to explain the fact that a scalar model made in all parts of the same material as the prototype will behave similarly to the prototype when rotated at speeds such that the centrifugal force is as many times the gravitational force as the prototype dimension is a similar linear model dimension. The reader is referred to earlier papers[1,2] for further information regarding these statements.

All models were made of sandstone and were as in Fig. 6. They are composed of a 3⁄16-in. ore body bed A, overlain by a 3⁄16-in. bed B, a 3⁄8-in. bed C and two 1⁄8-in. beds D and E. The model was rotated at ±2350 r.p.m. equivalent to a model ratio of 1268. It was worked back 1⁄4 in. at a time before being replaced in the centrifuge and again rotated. Table 1 gives model and prototype dimensions occurring in this discussion, the prototype dimensions being 1268 times the model dimensions.

Figs. 1, 2, 3 and 4 were taken while the model was rotating at 2350 r.p.m. One was taken for each face retreat of 26.4 ft., but for obvious reasons all are not presented.

Fig. 1 (*A–E*) shows the behavior of this type of structure with face retreats of 26.4 ft. and no artificial support.

Fig. 2(*A–E*) shows the behavior of this type of structure where breaker and face props were used. The props were exceptionally strong, being of steel 1⁄8 in. square in cross section, and capped with 1⁄16 in. ± of soft wood and paper, to allow for roof deflection. The face props were

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* Weighty structures in a centrifuge.

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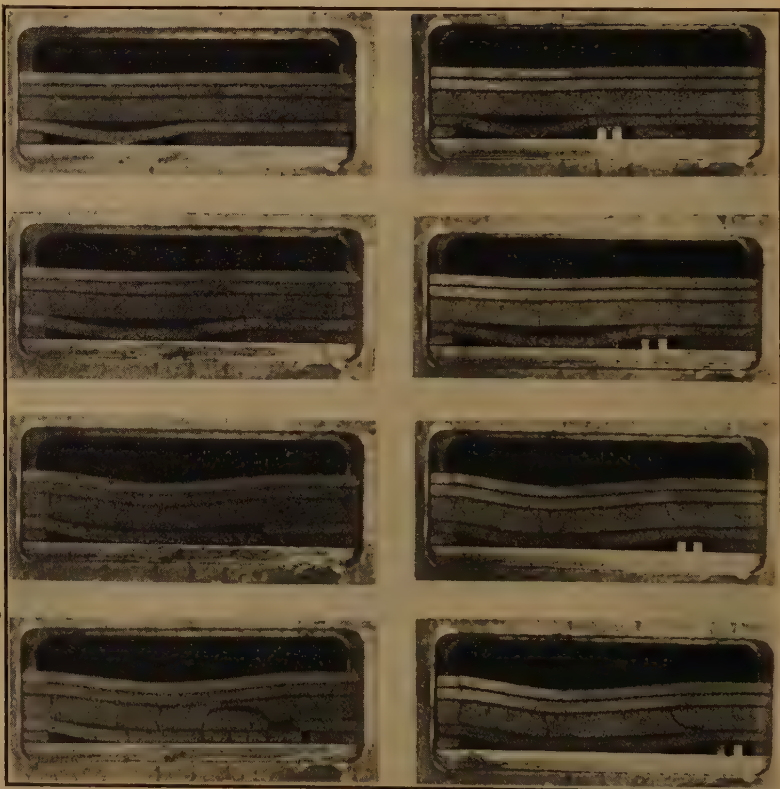
1 References are at the end of the paper.

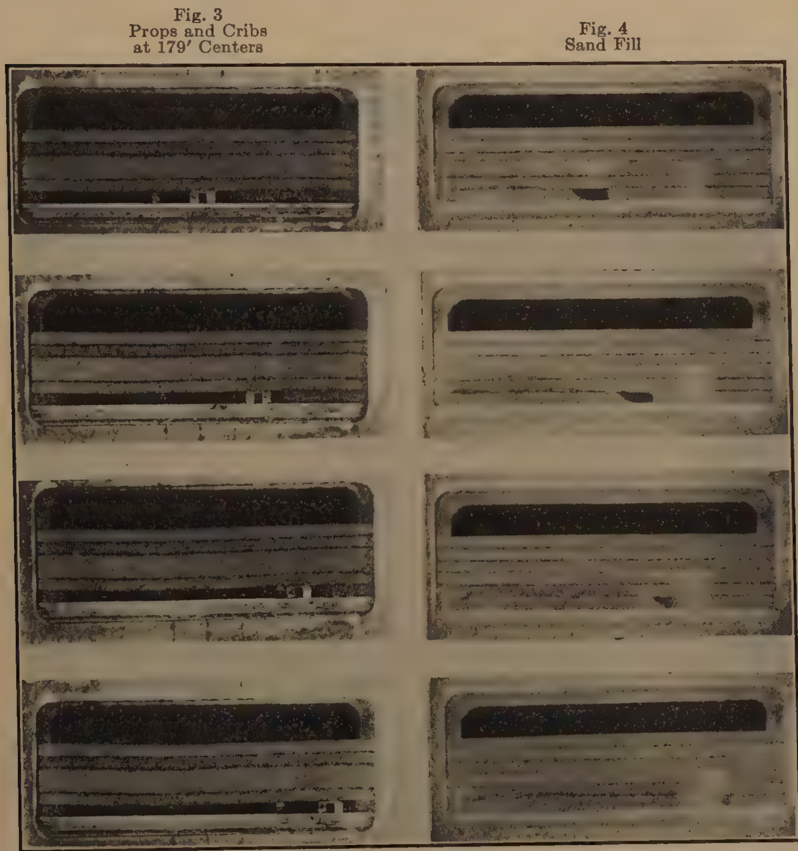
set 26.4 ft. (¼ in.) in front of the face and the breaker props were set ¼ in. (26.4 ft.) away from these. The laboratory procedure was to first mine out a ¼-in. section at the face with a hacksaw blade, remove the breaker props, and then replace them as face props. The model was then placed in the centrifuge and rotated at the calculated speed. In the field the procedure would be as follows: After the face had been cut,

Face Retreat to Center Line Breaker Prop		Fig. 1 No Support	Fig. 2 Breaker and Face Props
Model	Prototype		
A. 2.25″	238′		
C. 3.00″	317′		
D. 3.50″	370′		
E. 4.00″	423′		



shot, and loaded in a coal mine, or blasted and loaded in an ore mine for a distance uncovering a prop, a new face prop would be set. The breaker prop would then be pulled and used as a face prop for the next portion of face uncovered. The model procedure is similar to field procedure except that face working and prop movement take place out of the centrifuge; i.e., the roof, face, and props are not under stress. It is reasoned, however, that with careful retardation and acceleration of the centrifuge the additional stresses induced in model testing do not materially affect results. All face retreats noted in the figures refer to the distance from the stationary face to the breaker prop line.

Fig. 3(*A–E*) shows the behavior of this type of structure where cribs are used in addition to face and breaker props. The face and breaker props are the same as previously used and are placed the same distance apart. The face retreats referred to are from the stationary face to the breaker prop line. It was decided to place cribs at ¾ the span at which the underweight failed; i.e., ¾ by 238 ft., or 179 ft. Cribs are repre-



FIGS. 1–4.—FACE RETREAT WITH ARTIFICIAL SUPPORT. MODEL RATIO, 1268; 2350 R.P.M.

sented by wood sections ⅛ by 3⁄16 in. In the field the cribs would be ⅛ in. by $^{1268}/_{12}$, or 13.2 ft. wide.

Fig. 4(*A–E*) shows the behavior of this type of structure when sand filling was used. A section supporting a glass frame was attached to the model to keep the sand in (see Fig. 8). The procedure was to work the face ¼ in., or 26.4 ft., and place dry sand, allowing it to assume its angle of repose. The bottom was kept clear for 42 ft. in front of the face, which made the roof span; assuming a 45° angle of repose for the sand equal to 61.8 ft. No additional support of any kind was used.

Discussion of Results

No Support, and Breaker and Face Prop Series

The experimental results are presented in Table 2 and Figs. 1 and 2. It would be expected that these two series would behave similarly between the stationary and working face and between the stationary face and breaker prop line. The experimental results show that the first complete roof failure occurred with a 238-ft. retreat for both and that the

Table 1.—*Model and Prototype Dimensions*
Model Ratio, M.R. = 1268

Bed	Model, In.	Prototype, Ft.
Bed E, surface layer	3/16	19.8
Bed D, surface layer	3/16	19.8
Bed C, surface layer	3/8	39.6
Total overburden on roof B		79.2
Bed B, immediate roof	3/16	19.8
Total cover		99.0
Bed A, vein being worked	3/16	19.8
Total working face retreat	2.25	238
	2.50	264
	2.75	290
	3.00	317
	3.25	343
	3.50	370
	3.75	396
	4.00	423
	4.25	449
	4.50	475

additional retreat for the first crack to appear in the immediate roof B either over the working face or over the breaker prop line was also the same; i.e., 317 ft. − 238 ft., or 79 ft.

Rockfalls

In the no-support series rockfalls occur either at the working face or at a position behind the working face where a roof crack in bed B (Table 1) had previously shown. In the breaker and face-prop series, rockfalls never occur at the breaker prop line, but at a position behind the breaker prop line where a roof crack had previously been induced. A rockfall has never taken place either between the breaker and face props or between the face props and face. From the standpoint of safety, therefore, breaker and face props are desirable. They should be of sufficient strength and should be capped to allow for roof deflection. For economical operation with a fast-moving face they should also have the

TABLE 2.—*Summary of Observed Experimental Data*

Face or Breaker-line Retreat	1 No Support	2 Breaker and Face Props	3 Props and Cribs at 179-ft. Centers	4 Sand Fills
2.25 in. or 238 ft.	Fig. 1*A*. Bed B down with three breaks. Bed C has tension crack at roof center. Beds D and E cracked over stationary face.	Fig 2*A*. Bed B down with three breaks. Bed C has tension crack to right of roof center.	Fig. 3*A*. (N.O.C.)	Fig 4*A*. (N.O.C.)
2.5 in. or 264 ft.	Fig. not shown. Rock fall at working face.	Fig. not shown. Rock fall at previous breaker prop crack.	Fig. not shown. (N.O.C.)	Fig. not shown. (N.O.C.)
2.75 in. or 290 ft	Fig. not shown. (N.O.C.)	Fig. not shown. Bed C cracked over stationary face. Beds D and E cracked at center of original 2¼-in. span.	Fig not shown. (N.O.C.)	Fig. not shown. (N.O.C.)
3.00 in. or 317 ft.	Fig. 1*C*. Bed B cracked over working face. Central portion flat. Surface crack over stationary face.	Fig. 2*C*. Bed B cracked over breaker prop. Central portion flat surface crack over stationary face.	Fig. 3*C*. Bed B cracked at span center between stationary face and crib 1.	Fig. 4*C*. (N.O.C.)
3.25 in. or 343 ft.	Fig. not shown. Bed B cracked over working face. Rock fall under previous crack. Bed C cracked over working face.	Fig. not shown. Bed B cracked over working face. Rock fall under previous crack.	Fig. not shown. (N.O.C.)	Fig. not shown. (N.O.C.)
3.50 in. or 370 ft.	Fig. 1*D*. Bed B cracked over working face. Rock falls under previous crack. Bed C cracked over working and stationary faces. Beds D and E show double cracks near stationary face and at span center. Surface cracks over stationary and working face and double cracks at center.	Fig. 2*D*. Bed B cracked over breaker line. Rock fall under previous crack. Bed C cracked over breaker line and stationary face. Beds D and E show double cracks at center. Surface cracks over stationary face and 3¼-in. breaker line and double cracks at center.	Fig. 3*D*. Crib 2 added. (N.O.C.)	Fig. 4*D*. (N.O.C.)
3.75 in. or 396 ft.	Fig. not shown. Bed B cracked over working face.	Fig. not shown. Bed B cracked over breaker prop.	Fig. not shown. (N.O.C.)	Fig. not shown. (N.O.C.)
4.00 in. or 423 ft.	Fig. 1*E*. Rock fall at face. Bed B failed at 3¼-in. crack.	Fig. 2*E*. Bed B cracked over breaker prop.	Fig. 3*E*. Bed B cracked with rock fall between cribs 1 and 2.	Fig. 4*E*. (N.O.C.)
4.25 in. or 449 ft.	Fig. not shown. Finish.	Fig. not shown. Bed B cracked over breaker prop. Bed C, D and E cracked over breaker prop.	Fig. not shown. (N.O.C.)	Fig. not shown. (N.O.C.)
4.50 in. or 475 ft.	Finish.	Fig. not shown. Bed B cracked over breaker prop. Bed C cracked over breaker prop. Surface crack over breaker prop.	Fig. not shown. Bed C cracked over crib 1. All beds cracked over stationary face. Surface crack at stationary face.	Fig. not shown. (N.O.C.)
5.1 in. or 539 ft.	Finish.	Finish.	Fig. not shown. Beds B and C cracked at span center. A new crib is to be added.	Fig. not shown. (N.O.C.)

N.O.C. = no observable change.

property of being easily removed. This is the procedure generally followed in mining and it is interesting to note that practical experience has evolved this practice.

The results with props designed to allow the roof to sag or deflect may be considered as evidence that the calculations for prop size need be based only on supporting the underweight. The basis for calculation may be a continuous beam with three spans and the largest size prop thus calculated should be used for both face and breaker props.

Roof Cracks

In the no-support series tension cracks occurred in bed B (Table 1) over the working face for each 26.4 ft. of face advance, for face retreats between 317 and 423 ft. At 423 ft. a rockfall occurred at the face, thus stopping operations. In the breaker and face-prop series, tension cracks occurred in bed B over the breaker props for each 26.4 ft. advance of the breaker line, for breaker-line retreats between 317 and 475 ft. The retreat was stopped because the limits of visibility of the apparatus had been reached. In this series no rockfalls occurred that would tend to make operations unsafe. The conclusion from the preceding is that roof cracks in the immediate roof occur at the position of maximum support and eventually occur at regular intervals.

It may be well to consider another point of interest, especially for mines using systematic timbering without calculations as to size or placing. Experimental evidence points to the fact that roof breaks and rockfalls occur at sections of maximum support. Improper systematic timbering may therefore, under certain conditions, be the direct cause of a very treacherous roof, for timber lines are lines of maximum support.

Breaker and Face Prop Supports

In the series with breaker and face props (Fig. 2), bed C (Table 1) behaved in a somewhat different manner than in the no-support series. The first tension cracks with a 238-ft. breaker prop retreat occurred to the right of the span center and is explained by the fact that some roof deflection was allowed over the prop. It also cracked first over the stationary face with a 290-ft. face retreat. Other cracks in bed C occurred at face retreats of 370 ft., 449 ft. and 475 ft. At 370 ft., C also cracked over the stationary face.

Another feature of interest is *the type of crack that is obtained over supports that are solid as compared to those over yielding supports.* The cracks in bed C (Fig. 1, *A–E*) over solid supports tend to approach the vertical. The cracks that start over the props—yielding supports—are decidedly curved, with the approximate center of curvature at the top of the bed and toward the unworked portion (Fig. 2, *D* and *E*).

In the no-support series, bed C had its first crack in the working face after a 343-ft. face retreat. With a 370-ft. face retreat it cracked both

over the working and stationary face and arch caved as shown with a 423-ft. face retreat. Because of the type of break in bed C (Fig. 1*E* and Table 1) over the supports, and because of its inverted arch form, it appears evident that its deformed portion will exert a lesser upward thrust than bed C of Fig. 2*E*, whose crack shape over the yielding prop is such as to allow it to wedge itself between bed B and its standing portion.

Props and Cribs

The test results on this series are illustrated in Fig. 3 and Table 2. For this series cribs were placed at ¾ the span for failure, or at 179-ft. intervals from the stationary face. The cribs were of wood ⅛ by $^3/_{16}$ in., representing cribs 13.2 ft. wide in the field. The working face retreated 26.4 ft. after each run and the props moved as previously explained. When the breaker props were 205.4 ft. from the crib line a new crib was installed. The experimental results show no rockfalls at the face, or between breaker and face props. It is evidently safe for men and equipment at the face.

The first crack in the immediate roof B occurred after a breaker prop retreat of 317 ft. and consisted of a roof crack about 85 ft. from the stationary face. This is evidence that the No. 1 crib is supporting the underweight, for in the previous cases the roof failed completely after a 238-ft. face retreat. It may be observed, however, that the crib is not supporting the overweight, for bed B has sagged away from it. With a retreat of 4½ in. (475 ft.) crib 1 crushed askew, and crib 2 showed evidence of taking the load. It is now evident that crib 1 is loaded by the overweight. Cracks occur in all layers over the stationary face and a surface crack is also visible there. The first surface crack over the stationary face for the two previous series occurred after retreats of 317 ft. A tension crack occurred in bed C (Table 1) over crib 1, but no cracks of any kind have as yet occurred over the breaker props.

With a retreat of 5.1 in. (539 ft.) and now in a position to place a third crib, cracks occurred in beds B and C at the span center.

At this stage it may be interesting to note that the effects of the cribs were: (1) to increase the time or face retreat for the first surface crack to appear over the stationary face; (2) to do away with cracks over the breaker prop line. It may be reasoned, therefore, that cribs contribute to safety. The other side of the picture is that roof breaks always take place at or behind the breaker prop line and if the breaker props are pulled with the men between the face and face props, no danger to the men is present. Cribs, therefore, for these conditions do not contribute to the safety of a well-supervised mine and are an unnecessary expense.

Sand Filling

Fig. 4 shows the experimental results with sand filling. From previous experiments it was determined that the immediate roof failed at a

238-ft. span. It was then determined that with sand filling allowed to assume its normal angle of repose (45°), a 42-ft. floor span would give a 61.8-ft. roof span. Since the strength of a beam varies as the square of the span, the safety factor with this condition would be $\left(\frac{238}{61.8}\right)^2$ or 14+. The resulting tests substantiated these conclusions, as for this series no props were used; the face was worked back 26.4 ft. each time, and dry sand placed without tamping and allowed to assume its angle of repose. The light spots on the photographs are glare from the glass holding the sand in position (Fig. 8) and have no ill effect except in 4*E*, where it is difficult to see the working faces because of it.

It is interesting to note that a 61.8 ft. span was kept open in front of the working face while the face retreated 5.1 in. or 539 ft. During this retreat no cracks occurred in any bed and only slight subsidence was noted.

It is evident therefore that *sand* is ideal support material even when placed without tamping, and allowed to assume its angle of repose. It is also evident that the distance between face props and face, and face and breaker props may be increased to 61.8 ft. or more. The maximum span at the working face, important where mechanical equipment is to be used, may therefore be determined from model tests. If a safety factor of 10 is used in determining this value, it will be safe.

Subsidence

Surface Cracks

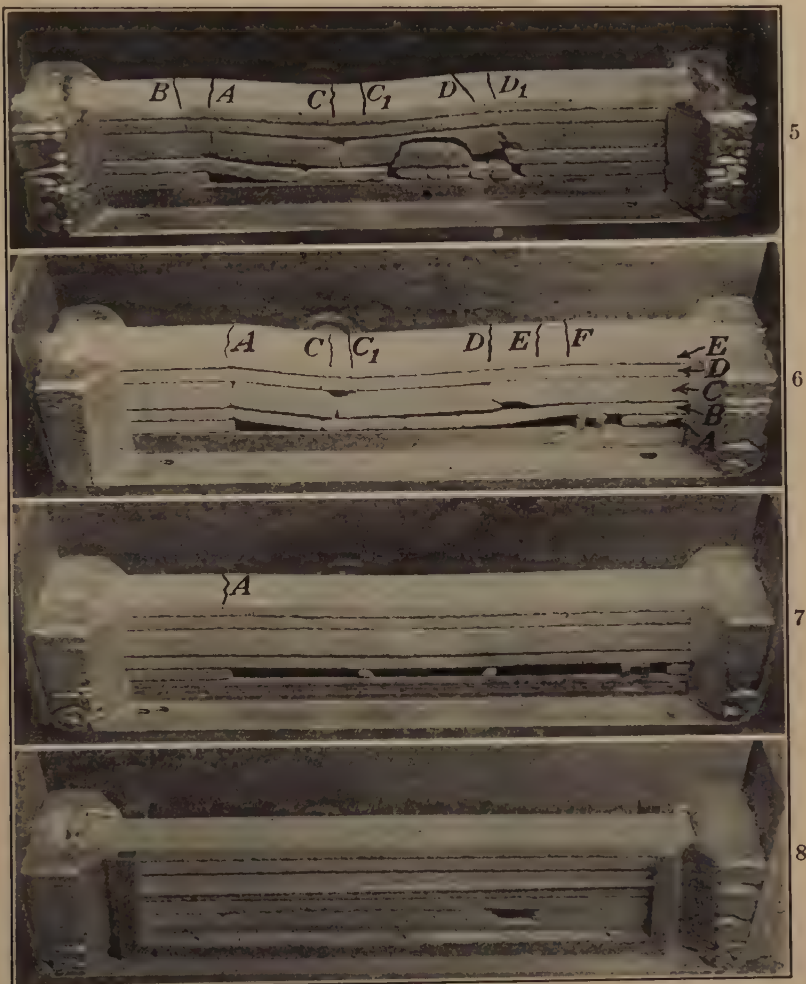
In the no-support series the first surface crack *A* (Fig. 5) appeared over the stationary face after a face retreat of 317 ft. In the breaker and face-prop series the first surface crack *A* (Fig. 6) appeared over the stationary face after a breaker-line retreat of 317 ft. In the prop and crib series the first surface crack *A* (Fig. 7) occurred after a breaker-line retreat of 475.

It is evident, therefore, that the presence of breaker and face props had no effect on the time or position of occurrence of the first surface crack. Cribs, however, had the effect of increasing the face retreat or time of occurrence of this surface crack but not its position.

In the no-support series (Fig. 5) additional face retreat to 370 ft. resulted in crack *B* over the stationary face, cracks *C* and C_1 at the center ½ in. apart and cracks *D* and D_1, 343 and 370 ft. respectively from the stationary face.

In the face and breaker-prop series (Fig. 6), breaker-line retreat to 370 ft. resulted in light cracks, *C* and C_1 at the center and crack *D* at 343 ft. from the stationary face. Light surface cracks also took place over the breaker line at retreats of 449 and 475 ft. (*E* and *F*). No succeeding surface cracks were obtained in the series with props and cribs. In the series with sand filling no surface cracks of any kind occurred.

The observed data in Table 2 and Figs. 5, 6, 7 and 8 are therefore evidences that the greater the amount of underground support the less tendency there is for the surface to crack. From the standpoint of mine drainage in regions where surface waters find their way through cracked



FIGS. 5–8.—MODELS IN HOLDERS.
Letters at side of Fig. 6 indicate: *A*, ore body; *B*, immediate roof; *C*, *D* and *E*, beds.

and fissured workings, there will be an economic balance between drainage and artificial support costs. No-support methods resulted in six surface cracks for a 370-ft. face retreat. Breaker and face-prop support methods resulted in four surface cracks for a 370-ft. breaker-line retreat. Props and crib-support methods resulted in only one surface crack for a 370-ft. breaker-line retreat, while sand-filling support methods resulted in no surface cracks for any retreat.

Since the mathematics of calculating the probable place of occurrence of future surface cracks for any of the procedures are too involved, their probable position is best determined by model experiment.

Surface Subsidence

The curves in Fig. 9 show surface subsidence for various face retreats and the methods of support investigated. Curve *S* represents the original surface; curve *A* the surface after a face retreat of 264 ft.; curve *B* the surface after a face retreat of 317 ft.; curve *C* the surface after a face retreat of 370 ft. and curve *D* the surface after a face retreat of 423 ft.

TABLE 3.—*Summary of Subsidence Data as Taken from Curves of Figure* 9

1 Description	2 Face or Prop-line Retreat, Ft.	3 Area of (+) Down Subsidence, Ft.	4 Maximum Vertical Subsidence, Ft.	5 Negative Subsidence Rise, Ft.	6 Area of (−) Negative Subsidence, Ft.
No support	264	285	3.17	0.36	195
	317	359	4.65	0.36	121
	370	450	17.55	0.048	30
	423	480	18.23	0.000	0.00
Breaker and face props	264	270	3.20	0.268	63
	317	317	5.42	0.00	0
	370	375	16.90	0.865	138
	423	420	17.85	0.576	50
Props and cribs	264	300	0.815	0.00	0
	317	317	1.92	0.24	130
	370	465	3.41	0.048	15
	423	480	4.48	0.00	0
Sand fill	264	201	0.720	0.336	264
	317	276	0.720	0.144	189
	370	380	0.480	0.00	0
	423	400	0.816	0.336	81

Observations were limited to an area represented by a line 450± ft. long and measured to the right of the stationary face. The curves were obtained from measurements on a photographic plate and are correct to 0.001 in. in the prototype. The accuracy here is therefore better than in the paper published in 1938.[3]

Table 3 is a summary of data taken from the curves in Fig. 9. Column 1 is self-explanatory. Column 2 is the result of a series of measurements between the stationary and working face or breaker-prop line multiplied by the model ratio (1268). Column 3 is a series of horizontal measurements from the stationary face to the intersection of the subsided surface with the original surface multiplied by the model ratio. Columns

4 and 5 are vertical measurements between the original and subsided surface at that section multiplied by the model ratio. Column 6 is the length of original surface above which the surface has risen.

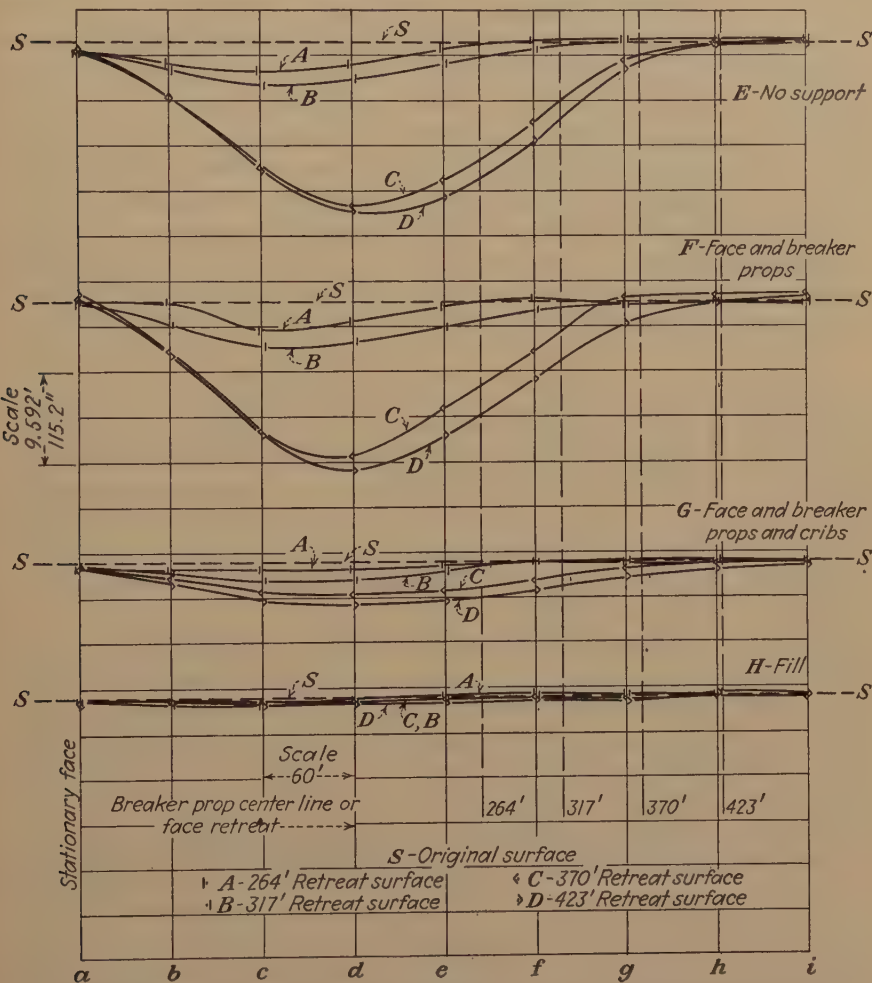


FIG. 9.—SURFACE SUBSIDENCE AS AFFECTED BY TYPE OF ARTIFICIAL SUPPORT.

Positive Subsidence

Table 3 and the curves in Fig. 9 show surface settlement called positive subsidence, and surface rise referred to as negative subsidence. It is interesting to note that the minimum positive subsidence was obtained where sand filling was used and was equal to 0.816 ft. The use of cribs and props limited the positive subsidence to 4.48 ft. but it is reasonable to believe that the time effects would increase this value considerably, depending upon the crib construction. With breaker and face props, the

amount of positive subsidence was 17.85 ft., while in the no-support series the amount of vertical subsidence was 18.23 ft. These laboratory results are substantiated by general field experience. They provide, in addition, quantitative relationships previously lacking.

All the areas affected by positive subsidence, except for the sand-fill series, are equal to or slightly greater than the worked-out area or the distance between the stationary face and breaker-prop line. It is a reasonable occurrence for these conditions, as the ore or coal near the working face is compressed by the load. Present laboratory measurements substantiate this fact. While the results with the sand fill do not quite substantiate this statement, the variation is probably due to the conforming supporting ability of the sand, or to an inability to obtain the correct point of intersection on the curves of Fig. 9*H* of the *S* with the subsidence curves *A*, *B*, *C* or *D*.

Negative Subsidence or Rise

The tendency for negative subsidence or rise of surface ahead of the working face or props is evidenced in the curves of Fig. 9 and column 5 of Table 3. It was observed in the field by C. A. Herbert and J. J. Rutledge[4] and others. The effect underground, where the coal apparently was relieved of part of its load at a given distance from the working face, is evidenced by data supplied by Charles T. Holland.[5] (Stations 1, 7 and 9 of his Figs. 9 and 10.)

A logical explanation to the writer as to the cause of the rise ahead of the working is the inverted arch action of a particularly strong, thick bed in the geological series overlying the ore body. One would, therefore, expect little rise where the geologic beds were thin and/or weak in shear or compression, or if support was so placed as to materially affect the curvature of the inverted arch.

In the no-support series (Fig. 1, *A–E*) nothing affecting the arch curve was introduced and the maximum rise was 0.36 ft., or 4.32 in. In the breaker and face-prop series (Fig. 2, *A–E*) yielding props were available, which did supply additional upward thrust on the arch and helped support it, the maximum rise was 0.865 ft., or 10.38 in. In the prop and crib series, where the formation of the inverted arch form was definitely interfered with, the maximum rise was 0.24 ft., or 1.68 in. In the sand-fill series, where the loose sand allowed some conformation and consequent inverted arching, the maximum rise was 0.336 ft. or 4.032 inches.

It is reasonable, therefore, to believe that some method of underground support may be devised to control the negative subsidence and its area. Where negative subsidence occurs generally in advance of the working face the load on the ore face is large as compared to the load on the ore or coal in advance of the face. When a thick bed bends, the beds overlying its convex portion are lifted and the load on the beds underlying the convex portion is decreased.

The relatively large load at the face is generally used to help break the ore or coal, as in the longwall fields of Illinois. If the ore or coal is undercut this breaking procedure is helped. The other effect of this large load at the face under certain geological conditions is to produce stresses on the ore face that may cause bursts. It is now evident that near the face there is a high stress area and that as one travels away from the face and in the solid the stress on the ore or coal is reduced considerably, especially under the area where negative subsidence has taken place.

Therefore if a drift is driven at right angles to a longwall face and the change in thickness or deformation of the ore bed determined as the face advances, an indication of the possibility of occurrence of a rock burst may be had. The reasoning is as follows:

1. Rock bursts occur at the working face if it is highly stressed.
2. From Hookes' law a measure of the stress in any portion is the amount of deformation there.
3. Therefore if in a drift or entry at right angles to the face, deformation gauges, as in Holland[5] or some other method, are placed and deformations measured, the stress in the ore bed near the face and beyond it may be approximated and provision made for either stopping work at the face or relieving the stress there introduced.

Summary and Conclusion

The barodynamic test procedure presented furnishes a simple, safe and inexpensive method for determining the facts on which conclusions may be based as to the desirability and effects of no support; breaker-prop and face-prop support; crib, breaker and face-prop support, or sand-fill support, in retreating longwall mining, for known geological and structural conditions. It therefore makes possible the substitution of fact for opinion as a basis for reaching conclusions that affect safety and operating costs. It provides in addition pertinent information on underweight and overweight behavior, surface subsidence, crack formation and methods of determining support requirements.

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DISCUSSION

[For discussion, see page 253.]

Photoelasticity and Its Application to Mine-pillar and Tunnel Problems*

By David Sinclair† and Philip B. Bucky,‡ Member A.I.M.E.

(New York Meeting, February 1940*)

The dimensions and shapes of mine structures may at present be determined by (1) field experience, (2) structural calculations, and (3) barodynamic tests.§ None of these, however, provide information as to the stresses, their direction and distribution in a mine structure. For the design of pillars, the placing of openings relative to working places such as gangways, haulage levels, etc., this information is desirable. Photoelastic methods provide this and, while relatively new, their application in all engineering fields is assuming increasing importance. This paper, therefore, will present the method with a few applications to mining, and with sufficient reference for further study by the reader.

PART I

The Photoelastic Method

The photoelastic method of stress measurement determines the stress distribution in a two-dimensional model by optical means. It makes use of the fact that when a piece of isotropic transparent material, such as glass, celluloid, or Bakelite, is stressed and viewed in polarized** light a picture is seen which indicates by its color bands of light, or fringes, the magnitude of the stresses, their direction and their distribution.

General Explanation

The stress at any point O in a structure (Fig. 1), may be resolved into two components, P and Q, each a pure tension or compression, at right angles to each other, called principal stresses.[1] To fully identify the stress, the values of P and Q and their directions or angle ϕ must be known.

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§ Weighty structures in a centrifuge.

** Vibrating in one direction.

[1] References are at the end of the paper.

The photoelastic apparatus (ref. 1, pp. 235–241 and ref. 2) (Fig. 3), provides a means of determining $P - Q$, the principal stress difference or twice the maximum shear stress at O, and the principal stress direction, angle ϕ. Fig. 4 shows the optical apparatus* for determining $P + Q$, the principal stress sum at O. Having two simultaneous equations with two unknowns P and Q, their values may be determined by algebraic or graphical means.

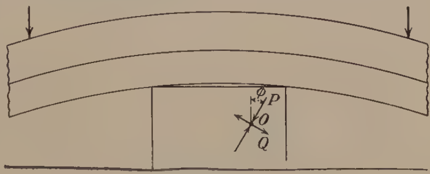


FIG. 1.—STRESSES AT A POINT.

P AND Q STRESSES

$P - Q$ Stress Values and Distribution.—To determine the $P - Q$ stress distribution under static loading the model is placed in a stress frame (Fig. 2) in position 5 of the photoelastic apparatus (Fig. 3) and loaded by a force on the vertical rod. Fig. 5*a* shows the $P - Q$ stress distribution in a 0.585-in. thick rectangular Bakelite pillar with such loading. Each fringe (dark area or line) is a series of points of equal maximum shear stress and therefore is a shear-stress *contour line*, termed an isochromatic. The value of the stress for each fringe is obtained experi-

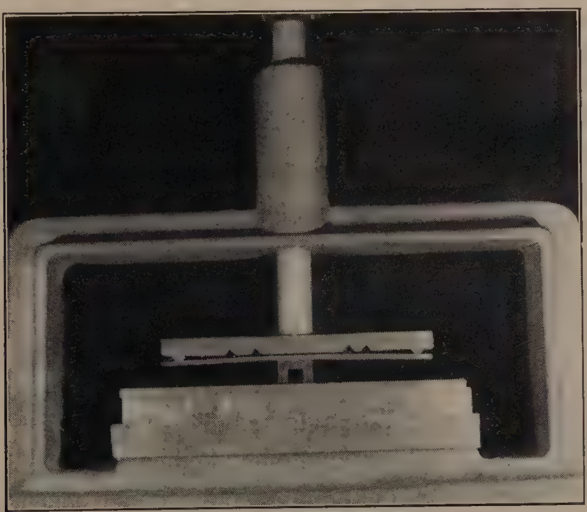


FIG. 2.—STRESS FRAME.

mentally by testing a calibrating beam in pure flexure (ref. 2, p. 182), the material being the same as that of the model. In Fig. 5*a* the zero shear fringe may be noted at the top center of the pillar, and one fringe corresponds to a maximum shear stress ($\frac{1}{2}(P - Q)$) of 78 lb. per sq. in. more or less than the previous one. By counting the fringes, the maximum shear

* A new method explained in part 2.

stress in the upper pillar corners is found to be 6 fringes = 468 lb. per sq. in. and in the lower corners 1 fringe = 78 lb. per sq. in. It is thus seen that portions of the model have low stress values while in others the values are high. The areas of relatively high stress in a pillar loaded

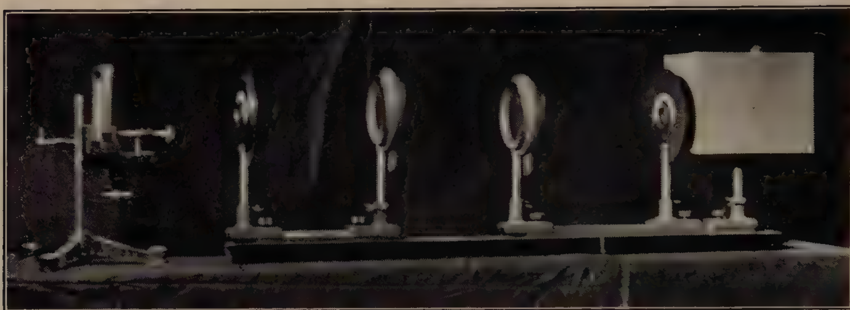


FIG. 3.—PHOTOELASTIC APPARATUS FOR DETERMINING $P - Q$ AND PRINCIPAL STRESS DIRECTIONS.

1. Source
2. Polarizer
3. Quarter-wave plate
4. Lens
5. Space for Model
6. Lens
7. Quarter-wave plate
8. Analyzer
9. Screen

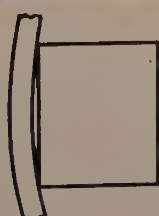
in this manner are small and it is logical to reason that failure will occur at these parts.

Direction of P and Q Stresses.—These are obtained by allowing the stressed model to remain in the same position and under the same load as for the determination of the $P - Q$ fringes (isochromatics). By

FIG. 4.—APPARATUS FOR DETERMINING $P + Q$.

removing the quarter-wave plates one obtains isoclinics; i.e., black lines or areas, which blot out the $P - Q$ fringes when polarizer and analyzer of Fig. 3 are oriented in the same direction as P and Q. Fig. 7b shows the 0° isoclinic of Fig. 7a obtained when polarizer and analyzer were vertical

FIG. 5.—ISOCHROMATICS AND ISOPACHICS IN PILLAR.



and horizontal. Comparing Figs. 7*a* and 7*b*, it is seen that the isochromatics in the lower and upper central portions are blacked out by the isoclinic, which means that the P stresses in the blacked out portions of the isochromatics 7*a* are vertical and the Q stresses are horizontal, or vice versa. The polarizer and analyzer may now be rotated together in the same direction 15° more or less at a time, so that the direction of the P and Q stresses in all parts of the model may be obtained (Fig. 7, *a* to *e*).

$P + Q$ Stress.—Fig. 5*c* shows the picture obtained with the stressed model of Fig. 5*a* in the apparatus of Fig. 4, which is shifted into position with the model under stress. The lines seen are interference fringes

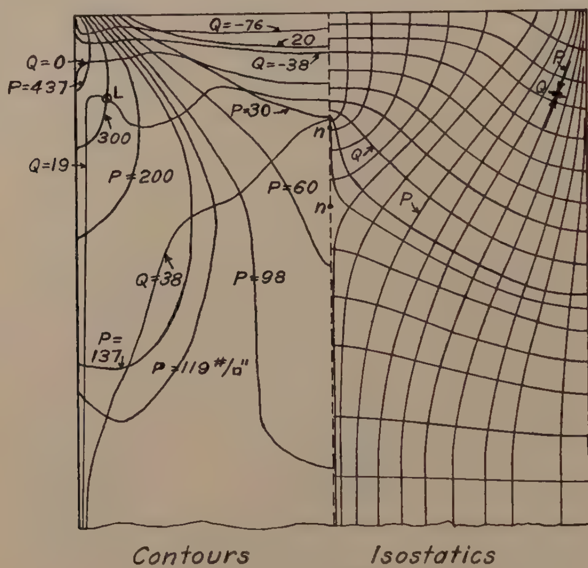


FIG. 6.—PRINCIPAL STRESSES IN PILLAR.

connecting points of equal model thickness. These are $P + Q$ contour lines and are called "isopachics." Fig. 5*b* shows the isopachics for the unstressed model. The value of the $P + Q$ stress for each fringe was found to be 75 lb. per sq. in. by testing a calibrating beam in pure flexure as for the isochromatics. It may also be obtained from the free boundary comparison of isopachics and isochromatics, since one principal stress along a free boundary is zero and the other parallels the boundary.

The isopachics were obtained by a new optical method developed by Sinclair, which is explained in detail in part 2.

Individual P and Q Stresses.—Fig. 6 shows the stress throughout the upper portion of the pillar. The magnitudes (ref. 2, p. 178) of the individual principal stresses are plotted on the left of the vertical center line and their directions (ref. 1, pp. 235–241) on the right. Compression is considered positive (+) and tension negative (−) with P always the

Fig. 7.—Full-load isochromatics and isoclinics.

algebraically greater stress.* To illustrate, at point L on the left $P = 300$ lb. per sq. in. and $Q = 19$ lb. per sq. in. acting in the direction of the heavy arrows shown at the corresponding point on the right.

Interpretation of Isochromatic $(P - Q)$ and Isopachic $(P + Q)$ Fringes

For determining the regions of a structure where overload or failure will first occur, or the parts that can be removed, the distribution of stress shown by the isochromatics and isopachics is usually sufficient. For example, consider the rectangular pillar loaded as in Fig. 5. The isochromatic pattern shows that the regions of greatest shear are at the corners of the pillar under the roof, since here the values of $P - Q$ are a maximum. If the prototype material is weaker in shear than compression, the pillar will first become overloaded at these upper corners. For geologic material usually weakest in tension, the isopachics should be considered. Inspection of Figs. 5*a* and 5*c* show that the isopachics have the same general form as the isochromatics, except at the top center. With this condition, $P - Q$ and $P + Q$ have approximately the same numerical value, which means that P or Q is small. In this model the greatest stress is compression, P, at the corners where both $P + Q$ and $P - Q$ are large. Only at the top center where both $P + Q$ and $P - Q$ are small does $P - Q$ differ appreciably from $P + Q$. Therefore, if there is any tension it will be so small that shear overload will be reached at the corners before tension overload occurs anywhere else. *Consequently, the regions of overload of pillars of geologic material may be obtained from the isochromatics alone.*

The stress analysis of a roof of this type of material, in which the tension stress is high, requires a study of the isopachics. Tension will be large where P or Q or both have large negative values. Thus large tension or overload will occur where an isopachic has a large negative value. One may also have large tension where $P + Q$ is small, due to a large positive value for P, i.e., compression, and a nearly equal negative value for Q, i.e., tension. In this region $P - Q$ will have a large value and the form of the isochromatics will differ widely from that of the isopachics. In this case tension overload will occur where the order of the isochromatic is large. It appears therefore that the point of overload can be determined for any type of material by a careful study of the isopachics and isochromatics without a knowledge of the individual P or Q stress values.

Various Conditions

Effect of Cutting Pillar Corners.—A pillar of approximately the same dimensions but loaded more heavily than in Fig. 5 gave the isochromatics

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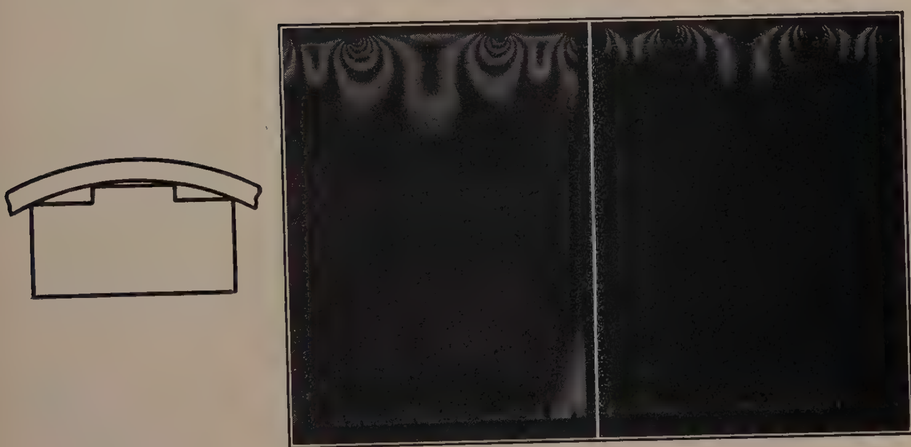
* The points marked n in Fig. 6, where $P = Q$, are known as "neutral" points.

and isoclinics shown in Fig. 7(*a-c*). The upper corners were then cut at an angle of 45° and the same load applied as in Fig. 7. The isochromatics are shown in Fig. 8*a*. The maximum shear stress is still 8 fringes = 624 lb. per sq. in. at the corners but there is no high stress concentration

a *b*

FIG. 8.



a *b*

FIG. 9.

FIG. 8 AND 9.—PILLAR SHAPES, STATIC LOADING.
a, isochromatics, full load.
b, 0° isoclinics, full load.

at the pillar edge as in 7*a*. Consequently, there will be less chance of pillar spalling. In the remainder of the pillar, unit stress values are low, and the 0° isoclinic (Fig. 8*b*) shows that the P and Q stresses act vertically and horizontally over most of the pillar.

Effect of Increasing Number of Roof and Pillar Contacts.—This effect could be obtained in a coal mine by making a top cut along the rib. The

section would then show as Fig. 9. For roofs that bend little, wedges or sand might be forced into this cut. In this case the cut was made thin enough so that the bending roof would rest on the pillar edges. It may be noted in Fig. 9*a* that the maximum shear stress at the center contacts is 5 fringes = 390 lb. per sq. in. and at the outer contacts it is 4 fringes = 312 lb. per sq. in. The spalling and crushing tendency at the corners has been considerably reduced here when compared to 7*a*, while the stress throughout the remainder of the block approximates that in 7*a* and 8*a*.

a b c d

FIG. 10.—PILLAR MINING, STATIC LOADING.
a, *b*, *c*, isochromatics ($P - Q$), full load.
d, 0° isoclinic of *c*.

If it may be reasoned that the tendency for pillar spalling or failure at a rib is a function of the stress concentration there, designs as in Figs. 8*a* and 9*a* will have less tendency in that direction than designs in 7*a*. The shape of 9*a* may be modified so that the stress concentrations at the center and end contacts may have other proportionate values, depending on the depth of cut and the support placed therein.

Effects of Openings in Pillars.—Fig. 10*a* shows the $P - Q$ stress distribution in a rectangular pillar. The maximum shear stress in the upper corners is 8 fringes = 625 lb. per sq. in. The stress at the bottom is about 1 fringe = 78 lb. per sq. in. On removing one-third of the material from the center portion, leaving two legs whose combined width is half the total width (Fig. 10*b*), the maximum shear stress in each leg is 2

fringes = 156 lb. per sq. in., twice what it was before, while the maximum shear stress at the upper corners has not been increased. If it is reasoned that the important point in pillar design is not to allow the maximum unit stress in the pillar at any point to equal or exceed the allowable unit stress in that material, one may mine still more of the pillar. This mining has not decreased the value of the pillar for support purposes.

Fig. 10*c* shows the $P - Q$ pattern with more material mined and the upper corners beveled at 45°. Here the legs are half as wide as those in 10*b* and the percentage of material mined is approximately 60 ±. The maximum shear stress is 8 fringes = 624 lb. per sq. in. at the corners and 4 fringes = 312 lb. per sq. in. in the legs. The 0° isoclinic (Fig. 10*d*) shows the principal stress direction in the legs to be vertical and horizontal. It is still as able to support its load as the solid one 10*a*. Therefore it is evident that a means is available for determining how much of a pillar may be mined.

Combined Photoelastic Barodynamic Tests

This method, previously described,[3] provides for structural loading by centrifugal means, which more nearly approximates field conditions

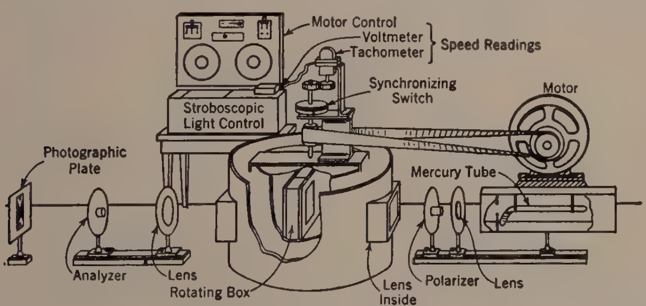


Fig. 11.—Diagrammatic representation of combined centrifugal and photoelastic equipment. (From *Civil Engineering*.[3])

and therefore gives a closer approximation to stress-distribution conditions in the field. Fig. 11 shows the combined centrifugal photoelastic apparatus and Fig. 12 is a model ready for test. The equipment is the same as that in Fig. 3 with the following exceptions. The stress frame, Fig. 2, is now replaced by the centrifuge, which is placed in position 5 of Fig. 3. The constant light source is replaced by a stroboscopic mercury arc, so that observations and photographs may be made while the model is rotating.

The results of these tests are shown in Figs. 13 to 17. The pillar is 1 × 1 × ½ in., while the roof not shown in the photographs is ¼ in. thick, both being of Bakelite.* The models were rotated at 2000 r.p.m.

* An additional lead load was applied to roof.

and the isochromatic and isoclinic photographs were taken at that speed. This rate of rotation is equivalent to a model ratio of 760 ± or a centrifugal field strength equal to 760 times gravity, which means that the model represents field prototypes of the same material, each of whose similar linear dimensions is 760 ± times those of the model.

Fig. 12 shows the method of loading and mounting the model and Fig. 13*a* the isochromatics or shear-stress distribution in a rectangular pillar. The maximum shear stress at the upper corners is 6 fringes = 468 lb. per sq. in., while along the lower half the maximum shear stress is 1 fringe = 78 lb. per sq. in. The 7½° isoclinic 13*b* shows that over most of the pillar the direction of action of the *P* and *Q* stresses are inclined 7½° to the horizontal. This inclination is due to a difference in span lengths on each side of the pillar.

Fig. 14(*a-b*) shows the effects of cutting the upper corners. The lengths of free pillar edges affected by high stress have been reduced.

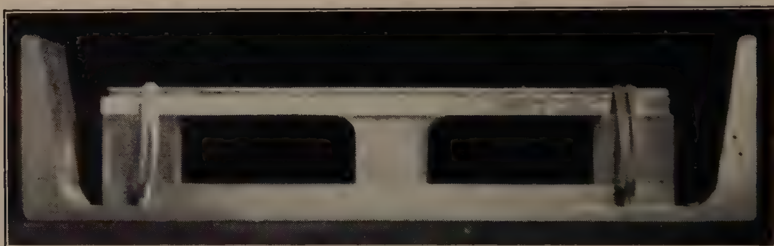


FIG. 12.—MODEL IN HOLDER.

The maximum shear stress still occurs at the upper corners and is 6 fringes = 468 lb. per sq. in., while the shear stress in the great portion of the pillar is 1 fringe = 78 lb. per sq. in. Throughout the great portion of the pillar, the 4° isoclinics (Fig. 14*b*) show that the *P* and *Q* stresses are inclined 4° to the horizontal and vertical. This is due to a lesser difference in span lengths on each side of the pillar than in Fig. 13.

Fig. 15*a* shows the effect of arching the pillar. The maximum shear stress has increased to 7 fringes = 546 lb. per sq. in. and a relatively large area along the curve is so stressed. The stress over the lower portion is approximately 1 fringe = 78 lb. per sq. in.

Fig. 15*b* shows the 0° isoclinic or the portions of 15*a* where the *P* and *Q* stresses are acting vertical and horizontal. The span on both sides of the pillar are therefore equal.

Comparison of Figs. 13, 14 and 15 provides evidence that better distribution of stresses along the pillar sides occurs when the upper corners are cut in. It was not felt necessary to shape a pillar as in Fig. 9(*a-b*) to be tested by centrifugal loading, since the general behavior as denoted by Figs. 13 and 14 closely approximated those of Figs. 7 and 8.

The statement made previously as to the desirability of making a top cut between pillar and roof as in Fig. 9 therefore holds.

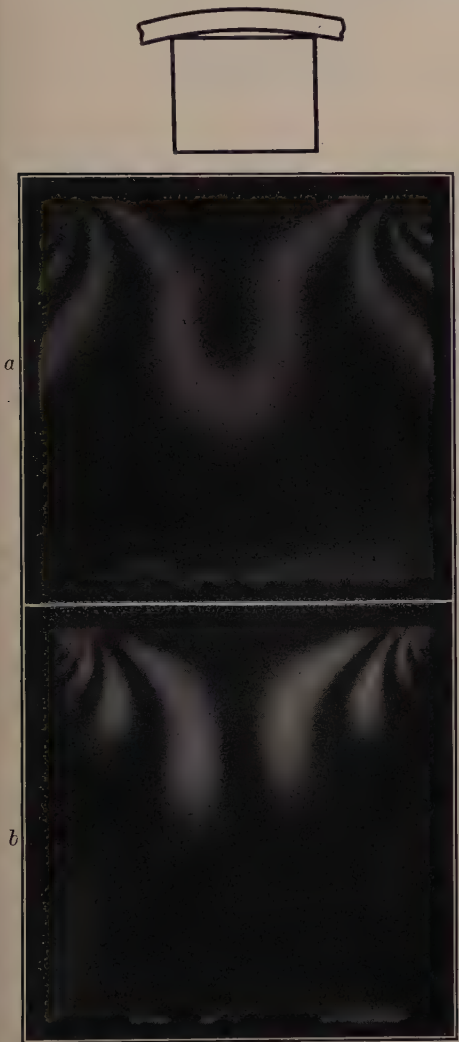


FIG. 13.

FIG. 14.

FIG. 13.—PILLAR SHAPE, CENTRIFUGAL LOADING, 2000 R.P.M.
a, isochromatic ($P - Q$), full load.
b, 7½° isoclinic, full load.

FIG. 14.—PILLAR SHAPE, CENTRIFUGAL LOADING, 2000 R.P.M.
a, isochromatic ($P - Q$), full load.
b, 4° isoclinic, full load.

Figs. 16 and 17 show the isochromatic patterns obtained when portions of the pillar are mined and the upper pillar corners are cut. In each of these cases, mining the center portions of the pillar does not affect the value of the pillar as a supporting medium, since the maximum shear

stress ($\frac{1}{2}(P - Q)$) at the corners is still 6 fringes = 468 lb. per sq. in. Further effects as to stress at ribs may be noted.

Stress Distribution under Pillars

The stress in a semi-infinite plane loaded by a uniform pressure over a small area has been calculated from the theory of elasticity (ref. 2, p.

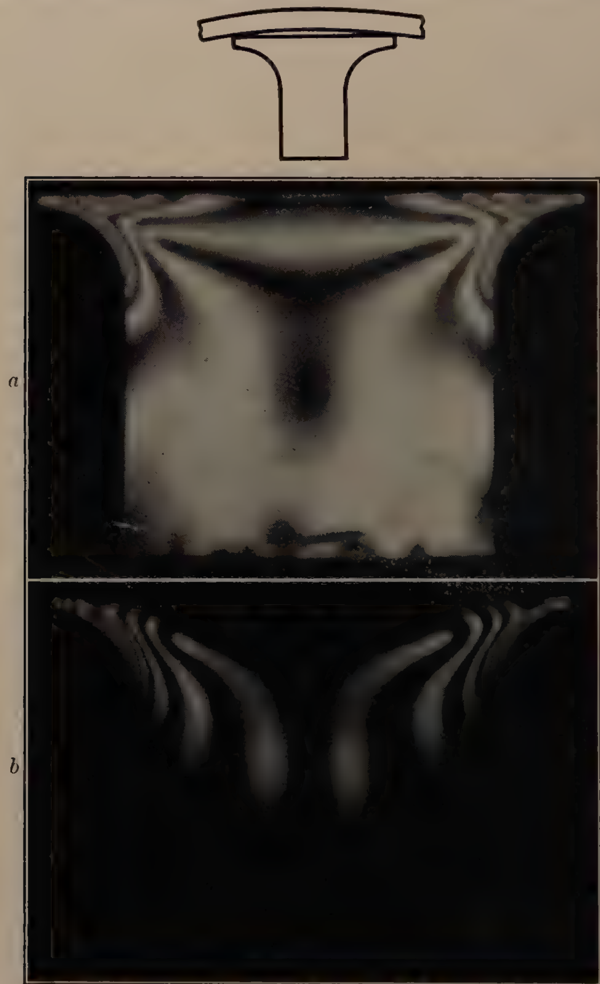


Fig. 15.—Pillar shape, centrifugal loading, 2000 r.p.m.
a, isochromatic ($P - Q$), full load.
b, 0° isoclinic, full load.

352). This condition approximately represents the stress in the bottom rock of Fig. 1 due to the weight of the pillar and the overlying material it supports. If b is the breadth of the pillar and p the uniform pressure exerted by it on the floor, then the maximum shear stress at any point O (Fig. 18) in the bottom rock is

$$\frac{1}{2}(P - Q) = \frac{p}{\pi} \sin \theta \qquad [1]$$

where θ is the angle subtended by the base of the pillar at O. The prin-

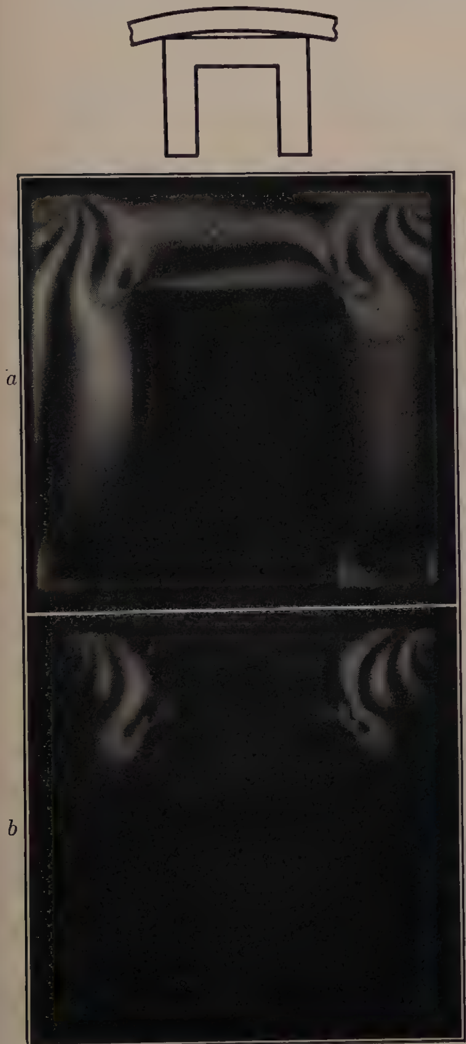


FIG. 16.

FIG. 17.

FIGS. 16 AND 17.—PILLAR MINING, CENTRIFUGAL LOADING, 2000 R.P.M.
a, isochromatics, full load.
b, 0° isoclinics, full load.

cipal stress sum at O is

$$P + Q = \frac{2p}{\pi}\theta \text{ (radians)} \qquad [2]$$

The isochromatics and isopachics are therefore circles passing through the corners of the pillar. The directions of the principal stresses are

along and perpendicular to the bisector of the angle θ. When $\theta < 30°$ the stress is approximately a pure compression directed toward the center, c, of the base of the pillar. The shear has its greatest value, $\frac{p}{\pi}$, along a circle whose diameter is the base of the pillar. It is zero along the base of the pillar and at a great distance from it. The principal stress sum, $P + Q$, has its greatest value, $2p$, at the base of the pillar and decreases to zero at a great distance from it. The stress is zero on the surface of the bottom rock on either side of the pillar. The rate of change in magnitude of stress is greatest at the pillar corners.

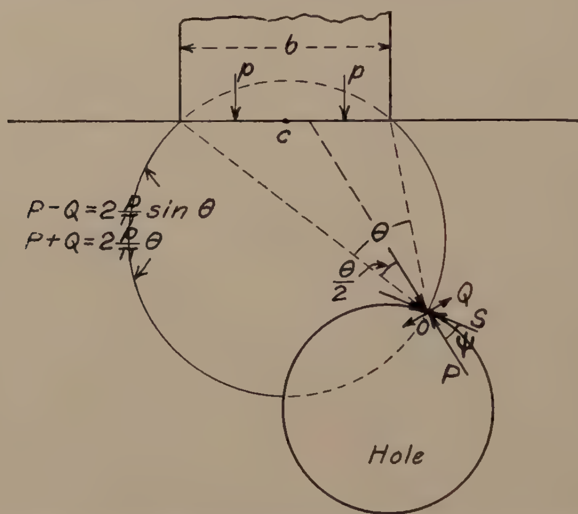


FIG. 18.—THEORETICAL STRESS DISTRIBUTION UNDER A PILLAR BEFORE AND AFTER A HOLE IS BORED.

The approximation to this state of stress is shown by the isochromatics (Fig. 19*a*), isopachics (Fig. 19*b*) and isoclinics (Fig. 20) obtained when a rectangular Bakelite block (1½ in. high, 3 in. long, and 0.305 in. thick) representing the bottom rock, was loaded by a rectangular Bakelite pillar (b = 0.253 in., depth = 0.305 in., p = 735 lb. per sq. in.). The isochromatics deviate from their circular form because of uneven and unsymmetrical loading and the finite size of the floor block. Most of the isopachics are circles and the isoclinics radiate approximately from the center of the pillar.

Stress Distribution around a Tunnel under a Pillar

The distribution of stress around the boundary of a horizontal circular hole in the bottom rock loaded as above can be approximately calculated from the theory (ref. 2, p. 481), but the distribution of stress in the material in the neighborhood of the hole has not been calculated. The

stress S at any point O (Fig. 18) on the boundary of the hole is a pure tension or compression tangent to the boundary, and of magnitude,

$$S = P'(4\cos^2\psi - 1) + Q'(4\sin^2\psi - 1) \qquad [3]$$
$$= S_P + S_Q \qquad [4]$$

Here P' and Q' are the principal stresses at O before the hole was drilled, obtained from eqs. 1 and 2, and ψ *is the angle between the tangent to the boundary of the hole at O and the direction of the principal stress P'.*

Fig. 21 shows S_P the part of S due to P', and S_Q the part of S due to Q' plotted against ψ around half the circumference. It is seen that $S = 0$ at four points* on the hole boundary, depending on the relative values of S_P and S_Q. Since P' is much greater than Q' throughout most of the floor, these points are located near $\psi = \pm 60$ and $\pm 120°$. When $\psi = 0°$ and $180°$, $S = 3P' - Q'$, a compression nearly three times as great as before the hole was drilled, and when $\psi = \pm 90°$, $S = 3Q' - P'$, a tension about equal in magnitude to the original compression.

FIG. 19.—ISOCHROMATICS AND ISOPACHICS UNDER A PILLAR.

Fig. 22*a* shows the isochromatics, Fig. 22*b* the isopachics, and Fig. 23 some of the isoclinics observed after the model of Fig. 19 had a 0.25-in. dia. hole drilled in the floor bottom, with its center 0.5 in. directly below the center *c* of the pillar.

The principal stresses in a section including the tunnel and the material up to the base are shown in Fig. 24. Here the magnitudes of the individual stresses are plotted on the left of the vertical center line and their directions on the right. Table 1 gives a comparison of some observed values at points on the tunnel periphery and the theoretical values, calculated from equations 1, 2 and 3, for the same points. The agreement is fair, the discrepancy being caused partly by initial stress

* These will be referred to as "zero" points.

TABLE 1.—*Observed and Calculated Tunnel Boundary Stresses at Points 1, 2, 3, 4, Fig.* 24

Point	ψ	θ	Average Observed S, Lb. per Sq. In.	Calculated S, Lb. per Sq. In.
1	90°	42°	−307	−283
2,4	0°,180°	30°	728	712
3	−90°	25°	−224	−192

due to heating while drilling the hole and partly by the relatively small size of floor block compared to size of hole and pillar. The ends of the

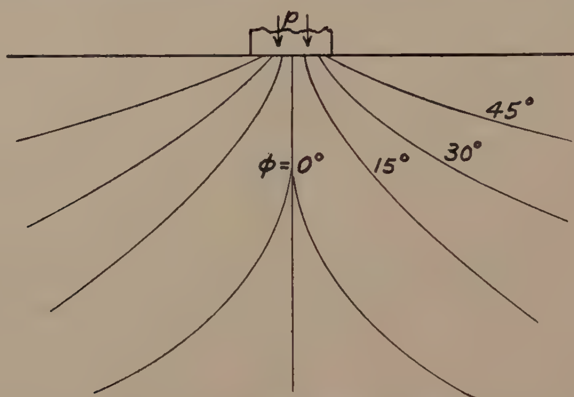


FIG. 20.—ISOCLINICS UNDER A PILLAR.

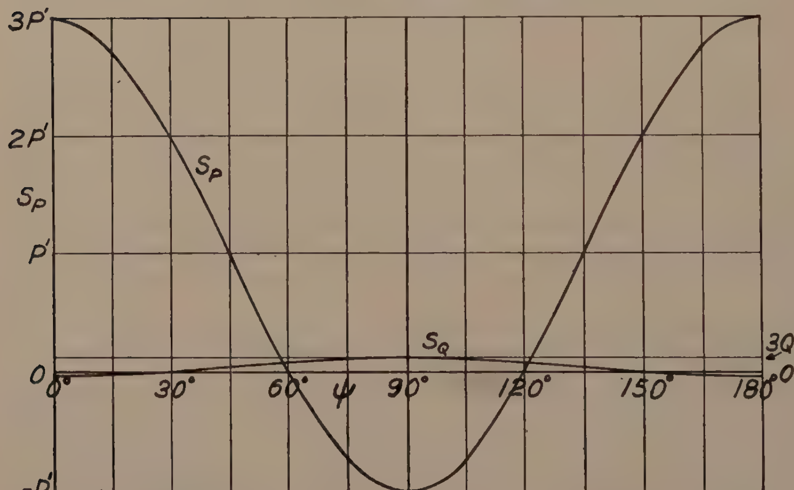


FIG. 21.—CIRCUMFERENTIAL STRESS $S = S_P + S_Q$ AROUND HOLE OF FIG. 18.

block tend to lift up from their support (ref. 2, p. 352) and a disproportionate amount of the load is borne by the middle portion of the block. In Figs. 19*b* and 22*b* the zeroth isopachic is found to be below rather than

at the surface, which is actually under a slight tension. In the field the bottom rock is restrained by other pillars or the orebody, so that the above case for which a theoretical solution can be obtained corresponds only approximately to an actual case, which requires the photoelastic method for its analysis.

The zero points (marked z in Figs. 24 and 22) are seen to be near $\psi = \pm 60°$ and $\psi = \pm 120°$. These were taken from the four points in Fig. 22b where $P + Q = 0$, which were found to correspond closely with the points in Fig. 22a where $P - Q = 0$. Between these points at the top and bottom of the hole, P is seen to be zero, and on the sides $Q = 0$. The line $Q = 0$ extends far out into the material.

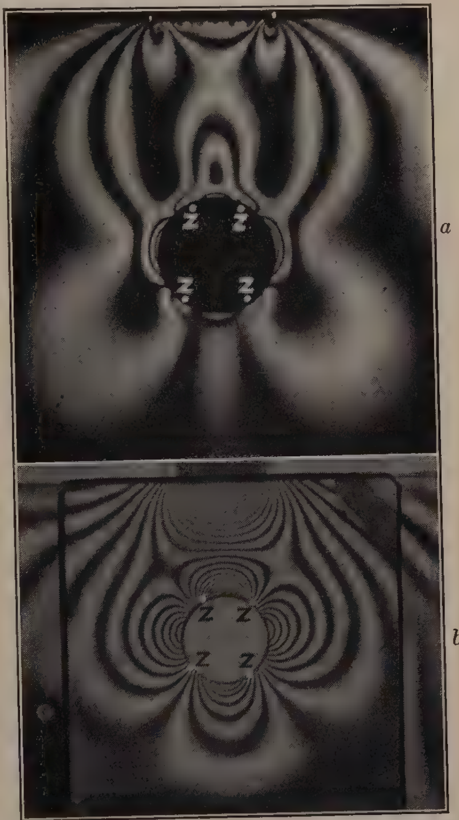


FIG. 22.—ISOCHROMATICS AND ISOPACHICS AROUND TUNNEL DIRECTLY BELOW PILLAR.

Near the sides of the tunnel at a distance of one-tenth the diameter of the tunnel from the wall, it is seen that the compression stress perpendicular to the wall is 113 lb. per sq. in. or about 15 per cent of p (735 lb. per sq. in.) the pressure exerted by the pillar at its base. In this region the compression parallel to the wall is 550 lb. per sq. in., or about 75 per cent of p. Near the top of the tunnel at a distance from the wall of one-fifth the tunnel diameter, there is a tension parallel to the wall equal to about 15 per cent of p, and a perpendicular compression of the same magnitude. It is seen from Table 1 that at the boundary near this region the observed parallel tension is 307 lb. per sq. in., or about 42 per cent of p. There would thus be a considerable tendency to develop cracks in this region as the tension approached the elastic limit of the rocks. There is also a region of somewhat less tension stress at the bottom of the tunnel.

TUNNEL TO ONE SIDE OF PILLARS

When the hole is drilled at the same distance below the pillar but off to one side, the magnitude of the stress is considerably less, as can be seen

from equations 1, 2, and 3 and a comparison of Figs. 22 and 25. The load in both cases is the same. Fig. 25 shows the isochromatics and iso-

FIG. 23.—ISOCLINICS OF FIG. 22*a*.

pachics around a hole drilled at the same distance below but off to the side of the pillar. Table 2 shows the theoretical and observed circumferential stresses in pounds per square inch at points 1, 2, 3 and 4, Fig.

TABLE 2.—*Observed and Calculated Tunnel Boundary Stresses at Points* 1, 2, 3 *and* 4, *Fig.* 25

Points	ψ	θ	Observed S, Lb. per Sq. In.	Calculated S, Lb. per Sq. In.
1	90°	18½°	−149	−153
4	180°	20°	438	490
2	0°	12½°	261	306
3	−90°	14°	−75	−118

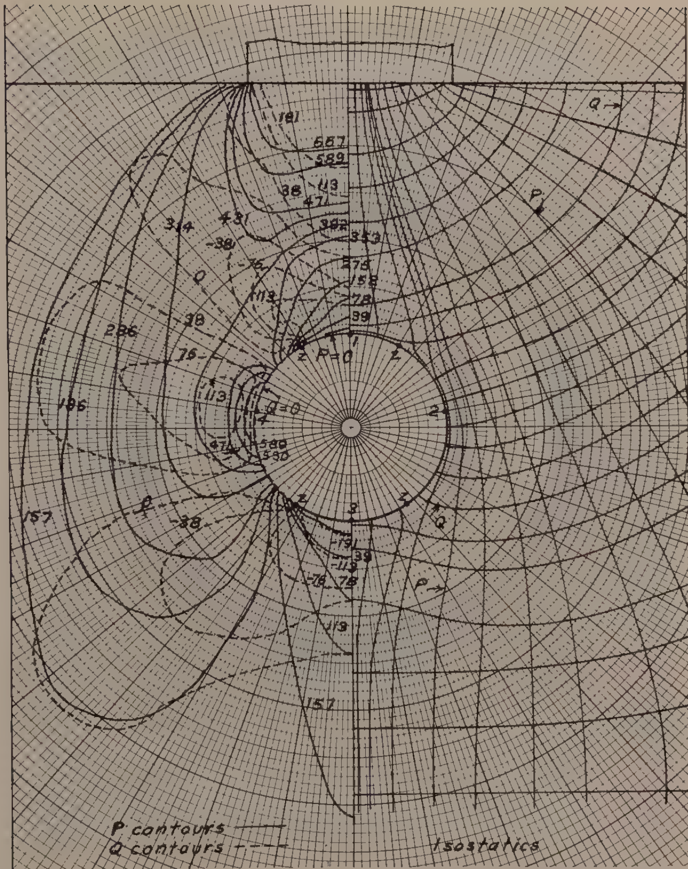


FIG. 24.—PRINCIPAL STRESSES AROUND TUNNEL.

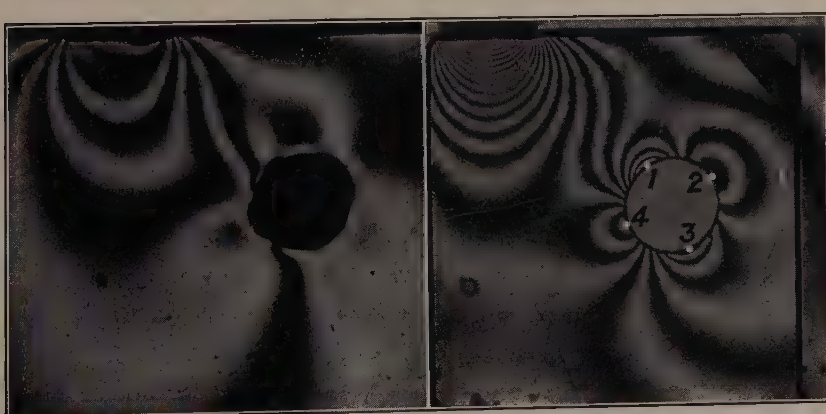


FIG. 25.—ISOCHROMATICS AND ISOPACHICS AROUND TUNNEL TO ONE SIDE BELOW PILLAR.

25. Here $p = 735$ lb. per sq. in., as before. Comparison with Table 1 shows that although the distance from pillar to tunnel has been increased by only 40 per cent, the stress has been reduced to one-half or even less of its former value. It should be noted that the regions of tension stress are now no longer directly overhead and underfoot but off to the side toward the pillar (above), and away from the pillar (below) at an angle of about 40° to the vertical. The zero points are now approximately overhead and underfoot and at both sides.

From equations 1, 2 and 3, it is seen that placing the tunnel nearer the floor at this same distance to the side would result in a further decrease in stress, although the distance from the pillar would be less. It is obvious, therefore, that openings should be placed in regions of low stress, which regions may now be determined. The stress distribution described is due to the pillar and the overweight supported by it, but neglecting the stress due to the weight of the material in the floor itself, and any lateral pressure. Usually the load supported by a pillar is large compared to the body weight of the floor on which it rests, and, if the stope is large the lateral pressure will also be relatively small. However, at great depths or with weak material, the self weight may become important and in some cases appreciable lateral stresses may remain in the floor even after the load it supported has been entirely removed.

The stress distribution around a horizontal circular tunnel in a gravitating body has been calculated from the theory of elasticity. Van Iterson, assuming the initial pressure before the tunnel is made to be hydrostatic—i.e., the lateral compression equal to the vertical compression—finds that the circumferential stress is everywhere a compression, being least in the roof of the tunnel. A similar result, based on the same assumption, is obtained by Sugihara and Sezawa[5] for vertical and inclined circular shafts. This state of initial stress may exist at great depths, and it is known[6] that the circumferential stress in this case is everywhere a compression equal to twice the initial compression. It is unlikely, however, that hydrostatic pressure exists anywhere near the surface. Hudspeth and Phillips[7] consider that the lateral pressure in a large horizontal bed before mining is a compression just sufficient to prevent lateral deformation by the vertical pressure. Mindlin[8] shows the effect on the tunnel boundary stress of variation of the initial lateral compression from equality with the vertical compression to zero. At the first extreme the values of circumferential compression agree roughly with those calculated from van Iterson's formula. As the lateral compression decreases the circumferential compression in the roof and floor decreases rapidly to zero and soon becomes a tension. With an initial lateral compression of zero the tension in the roof and floor is about half the magnitude of the compression at the sides. This is approximately the state of initial stress not too near the pillar of Fig. 18; i.e., where Q is relatively small.

It is proposed to investigate barodynamically the actual magnitude of the initial side pressure under various conditions by means of a high-speed centrifuge.

Relationships between the Results on Model and Effect on Prototype

Centrifugally Loaded Model.—The laws of similitude that enable one to calculate the effects on the prototype from the results on a model centrifugally loaded may be stated as follows:

$$MR = \frac{E_p \times C_g \times d_b}{E_b \times d_p} \tag{5}$$

and

$$s_p = s_b \frac{E_p}{E_b} \tag{6}$$

E_p = elastic modulus of prototype material.
E_b = elastic modulus of model material.
C_g = ratio of strength of centrifugal to gravitational field.
d_p = density of prototype material.
d_b = density model material.

$$MR = \text{model ratio} = \frac{\text{Prototype width or height dimension}}{\text{Similar linear model dimension}}$$

s_b = value of one fringe or stress at a point in model.
s_p = value of one fringe or stress at a similar point in prototype.

To illustrate: Assume prototype material for which $E_p = 10^6$, $d_p = 3$. For the model material $d_b = 1.5$, $E_b = 2 \times 10^5$. Assume also that the model is that of Fig. 13 and run at such a speed that $C_g = 760$. Then from eq. 5,

$$MR = \frac{10^6 \times 760 \times 1.5}{2 \times 10^5 \times 3} = 1900$$

i.e., the model dimensions of Fig. 13 must be multiplied by 1900 to get the prototype dimensions.

$$\text{From eq. 6, } s_p = 78 \times \frac{10^6}{2 \times 10^5} = 390 \text{ lb. per sq. in.}$$

or the value of the stress in a prototype whose similar linear dimensions are 1900 times that of the Bakelite model, where No. 1 fringe is visible is 390 lb. per sq. in. The stress in the prototype at the corners under the roof would be 6 × 390 lb., or 2340 lb. per sq. in.

Statically Loaded Model.—For photoelastic models built to scale and statically loaded, as, for example, the pillar over the tunnel in the bottom rock, Figs. 18 to 24, the following relationships hold:

$$S_p = \frac{MR \times S_b}{T_b} \qquad [7]$$

$$s_b = s_p = \text{value of one fringe in model and prototype} \qquad [8]$$

S_b = total static load on model, lb.
S_p = static load on prototype for full width and per inch length
T_b = model thickness, in.
Other values are as previously given.

To illustrate for the model of Figs. 22 to 24: Assume $MR = 480$. Then from data page 238

$$S_b = 735 \times 0.253 \times 0.305 = 56\tfrac{1}{2} \text{ lb.}$$

and from eq. 7 $S_p = \dfrac{480 \times 56\frac{1}{2}}{0.305} = 89{,}000$ lb. = the load on prototype.

The prototype dimensions are as follows:

$$\text{Pillar width} = \frac{480 \times 0.253 \text{ in.}}{12} = 10.12 \text{ ft.}$$

$$\text{Tunnel diameter} = \frac{480 \times 0.25 \text{ in.}}{25} = 10.00 \text{ ft.}$$

$$\text{Distance from center of tunnel to floor} = \frac{480 \times 0.5 \text{ in.}}{12} = 20.0 \text{ ft.}$$

From eq. 8, the stress at similar points in prototype and model are the same; they are listed in Table 1. This table indicates that tensile stresses (negative) of 300 lb. per sq. in. occur, which means tunnel failure if the prototype rock will not stand them.

Summary and Conclusion

It has been shown that: (1) the stresses in a mine pillar vary greatly and are influenced to a considerable degree by the load, the deflection of the strata over it and its shape, and (2) the stresses in openings and the intervening strata under pillars vary greatly and are determined by the pillar load and the relative positions of opening and pillar bottom. Combined barodynamic photoelastic methods offer a means of determining the values of these stresses, their direction and distribution. The application of these methods therefore should result in safer workings, and lower maintenance costs.

PART 2

Method of Obtaining $P + Q$

When a model under plane stress is viewed at right angles to the plane of the model, the following photoelastic or stress-optical relationships hold (ref. 2; ref. 1, p. 230):

1. The directions of polarization in the induced doubly refracting crystal lie along the mutually perpendicular directions of principal stress, P and Q.

2. The relative retardation, ∂, of two plane-polarized rays whose vibrations are parallel to P and Q, respectively, is directly proportional to the difference between the principal stresses $(P - Q)$ and to the thickness t of the model at the point through which the rays pass; i.e.,

$$\partial = (P - Q)tC \text{ where } C \text{ is a constant.}$$

3. The change in index of refraction for each of these polarized rays is a linear function of the principal stresses P and Q; i.e.,

$$\Delta u_P = C_1P + C_2Q$$
$$\Delta u_Q = C_1Q + C_2P$$

where Δu_P is the change in the index of refraction for the ray vibrating parallel to the principal stress P, Δu_Q the change in index of refraction for the ray vibrating parallel to the principal stress Q, and C_1 and C_2 are constants.

The following stress relationship also holds (ref. 2; ref. 1, p. 276):

4. The change of thickness Δt at this same point of the model is directly proportional to the sum of the principal stresses $(P + Q)$, and to the thickness of the model; i.e.,

$$t = -(P + Q)t\frac{n}{E}$$

where E is Young's modulus and n Poisson's ratio.

From rel. 1 can be obtained the directions of principal stress throughout the model and from rel. 2 the magnitude of $P - Q$, the principal stress difference. From eqs. 3 and 4 the magnitude of the principal stress sum can be obtained as follows.

A beam of unpolarized light incident at any point of the stressed model is broken up into two plane-polarized rays, vibrating according to rel. 1 in the direction of P and Q, respectively. The optical thickness of the unstressed model is ut for all rays, where u is the original index of refraction for the unstressed model. The optical thickness of the stressed model at the point considered is $(u + \Delta u_P)(t + \Delta t)$ for the ray vibrating in the direction of P, and $(u + \Delta u_Q)(t + \Delta t)$ for the ray vibrating in the direction of Q. Therefore the change in optical thickness or retardation due to the stress is:

$$d_P = (u + \Delta u_P)(t + \Delta t) - ut$$
$$d_Q = (u + \Delta u_Q)(t + \Delta t) - ut$$

Substituting the values given in rels. 3 and 4 and neglecting the negligibly small quantities $\Delta u_P \Delta t$ and $\Delta u_Q \Delta t$ one obtains.

$$d_P = -u(P + Q)t\frac{n}{E} + (C_1P + C_2Q)t$$

$$d_Q = -u(P + Q)t\frac{n}{E} + (C_1Q + C_2P)t$$

Thus, the beam emerging from the model at any point is composed of two plane-polarized parts having different retardations. The two parts of the beam are given the same retardation, as follows:

After emergence from the model, the beam is passed at normal incidence through a plane parallel crystalline quartz plate cut perpendicular to the optic axis and 0.139 in. thick.* This rotates all the directions of polarization through 90° at every point of the model. The beam is then passed through a second model stressed similarly to the first, whence it emerges with the two plane-polarized parts retarded equally, since the part of the beam that was polarized in the direction of P and retarded the amount d_P in the first model is rotated into the direction of Q by the quartz plate and retarded the amount d_Q in the second model, and vice versa. Therefore at every point of the model the two parts of the beam finally emerge with the same retardation, namely:

$$\begin{aligned} d_P + d_Q &= -2u(P + Q)t\frac{n}{E} + (C_1 + C_2)(P + Q)t \\ &= (P + Q)t\left(-2u\frac{n}{E} + C_1 + C_2\right) \end{aligned}$$

and the retardation is proportional to $P + Q$ since t and the quantities in the parenthesis are constants for any one material and wave length.

Apparatus for Determining $P + Q$ Values

For the observation of the principal stress sum, a Mach-Zehnder or Jamin interferometer (ref. 2, p. 85; ref. 1, p. 278) is used as shown in Fig. 26 and Fig. 4. A beam of accurately parallel rays of unpolarized monochromatic light strikes the first "half" mirror and is divided into two approximately equal parts, one of which is transmitted and sent through the double model and the quartz plate, the other reflected at about a right angle and sent through a compensating† quartz plate exactly like the first. The two parts of the beam are reunited at the second half mirror and sent through the collecting lens to the viewing screen. The mirrors must be placed so that they are approximately parallel and so that the optical distance traveled by the two beams is nearly the same for each; i.e., distance $1 + 3 = 2(u - 1)t +$ distance

* For large models the quartz plate may be made three times as thick for rigidity. It will then rotate the planes of polarization through 270°.

† If circularly polarized light is incident on the interferometer no compensator will be necessary.

2 + 4, Fig. 26. This can readily be accomplished by moving full mirror No. 1 to the right until distance 3 is equal to $2(u - 1)t$ + distance 2, and then turning both mirrors No. 1 in a counterclockwise direction until the fringes reappear.

In order that the mirrors may be lined up, they are mounted in holders having three adjusting screws, which permit the mirrors to be rotated through a small angle about a horizontal and a vertical axis. These also allow the mirrors to be moved bodily a short distance in a direction perpendicular to their own plane. The lining up is readily done by placing two cross wires in the beam between the source and the first half mirror, one near the source and the other near the mirror. When the mirrors are not properly aligned the collecting lens will form a double image of each cross wire, one for each of the two beams formed by the interferometer. Since the two cross wires have been placed at widely different distances from the collector, their images will be formed a

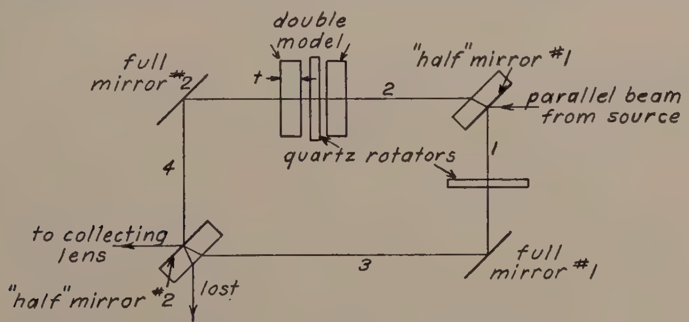


FIG. 26.—PLAN OF APPARATUS FOR DETERMINING $P + Q$.

considerable distance apart, so that it is convenient for this test to view the image with a telescope that can be readily focused. A little practice will show which way to move the mirrors in order to render the double image single. When both images of the cross wires appear single, the two parts of the divided beam must be nearly coincident and parallel after leaving the second half mirror. When this condition is reached the beams are in a position to interfere, and interference fringes will appear.

With a good parallel beam and well made mirrors and models, the whole model will be striped with straight, parallel and equidistant black and green fringes. Their number and direction will depend upon the relative tilt of the mirrors, as well as the condition of the model. Before the load is applied, the mirrors should be adjusted so that the model is covered as nearly as possible by one fringe.

If the load is now applied, a number of variously curved fringes will appear, depending on the distribution of stress in the model. These fringes are contours of path difference d between the beam that passed through the model and the one that did not. Since the path difference

was constant over the unloaded model, the path difference over the loaded model must be equal to a constant plus the retardation due to the stress; i.e.,

$$d = d_P + d_Q + \text{constant}$$

Since it has been shown that $d_P + d_Q$ is directly proportional to $P + Q$, these fringes are $P + Q$ contours, or isopachics.

The $P + Q$ fringe-stress value is obtained by observing the isopachics formed in the same beam model that was used to determine the $P - Q$ fringe-stress value. Fig. 27b (right half) shows the isopachics of this

a

b

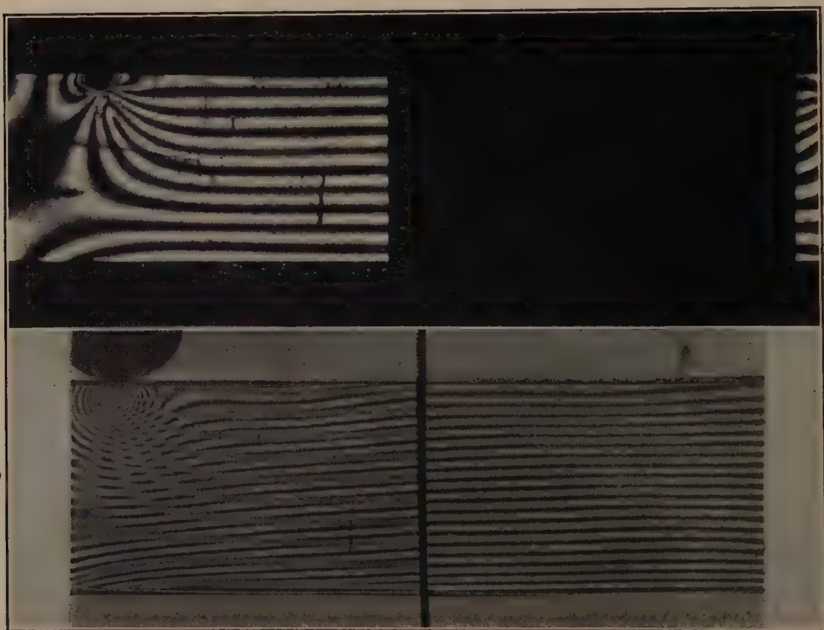


FIG. 27.—ISOCHROMATICS AND ISOPACHICS OF BEAM IN PURE FLEXURE.

beam stressed in pure flexure. (The left half of Fig. 27b shows the two overlapping sets of fringes observed where the beam did not pass through the quartz rotator.)

It is interesting to observe the elimination of the isochromatics caused by the introduction of the quartz plate rotator between the two models. The relative retardation, ∂, of rel. 2 is here zero, since both beams have been retarded the amount $d_P + d_Q$. Fig. 27a (right half) shows the elimination of the isochromatics of the beam in pure flexure and Fig. 28 the same effect around the hole of Fig. 22. This observation provides a criterion for similar loading of the two models as well as a check of the theory.

Discussion of Method

This method of obtaining the value of $P + Q$ has the following advantages and disadvantages:

Advantages

1. It enables the magnitude of $P + Q$ to be obtained over the whole model at once, as in Frocht's method,[9] rather than point by point (ref. 1, pp. 276 and 278).

2. The same model is used under identical loading conditions to obtain the magnitude of both $P - Q$ and $P + Q$ with little lapse of time between the two observations. In this laboratory all that has to be done

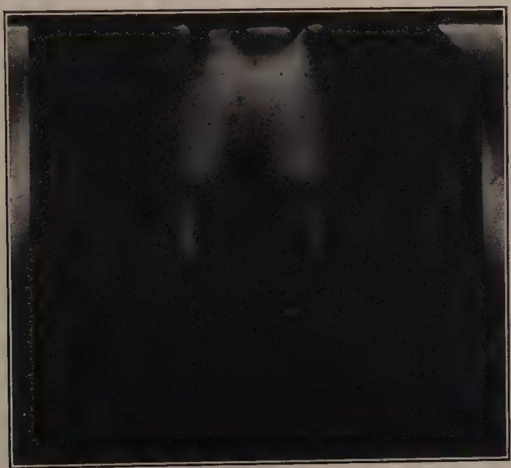


Fig. 28.—Elimination of isochromatics by quartz rotator.

between the observation of the isochromatics and isopachics is to remove the polarizer and slide the interferometer and quartz rotator into place.

3. The isopachics are observed with the model and loading machine at almost any convenient distance from the rest of the apparatus (except for the quartz rotator, which is independently supported between the models).

4. It is unnecessary to compensate for motion of the plane of symmetry[9] of the model. In the present transmission method, motion of the plane of symmetry causes no more change in the form of the isopachics than it does in the form of the isochromatics. In general the transmission method is relatively insensitive to distortion and displacement of the model, or vibration of the apparatus.

5. Once the interferometer is set up and adjusted it is ready for the observation of any model of roughly the same thickness, since only a slight adjustment of two mirrors is necessary thereafter.

Disadvantages

1. All models must be made in duplicate with their surfaces polished more carefully than is necessary for the observation of isochromatics. However, the models need not be plane sided, since in the polishing process the convexity of one or more surfaces can be made to compensate for the concavity of the others. Also, the usually unnoticeable wedge shape of the models can be entirely compensated for by adjustment of the interferometer mirrors. Furthermore, even if two or three fringes appear in the unloaded model it can frequently be used, since it is usually possible to shift the fringes to regions of the model in which one is not interested.

2. The two models must be made alike and loaded in the same manner. In practice this has not been found difficult to do, since, according to the principle of Saint-Venant (ref. 6, p. 31), the stress distribution will be the same throughout most of a model even though the distribution of stress near the point of application of the load be varied by small amounts. This is illustrated in Fig. 28, which shows that throughout most of the floor the isochromatics are completely eliminated, while just under the pillar they are partly visible.

3. The metallic mirrors cause some elliptical polarization of the light, which might distort the isopachics. Observations of the isopachics through a polarizer in different orientations showed a slight change in their intensity but no change in their form when the models were loaded properly, indicating that the effect of the polarization by the mirrors is negligible.

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DISCUSSION

[This discussion refers also to the paper by P. B. Bucky and R. V. Taborelli, beginning on page 211.]

C. A. Peterson,* Scranton, Pa.—Photoelasticity is entirely new and barodynamics is relatively so new as applied to mining that a discussion of this paper can properly be centered on the question of whether or not these methods can be profitably applied to actual mining problems. I shall, therefore, limit myself to attempting to answer that question as well as I am able.

Photoelasticity is an ingenious and fruitful, yet perfectly logical refinement upon earlier methods of observing barodynamic phenomena. Inasmuch as it has proved to be a most effective and practical aid to the solution of numerous problems of industrial design of a nature similar to many mining problems, there is no apparent reason why it should not be equally so in making barodynamic observations. Whereas formerly Professor Bucky was forced to limit himself to observation of flexure and failure as they manifested themselves on the exposed surfaces of his models, now he is able to look within the model and observe when, where and how the stresses develop in response to various types and degrees of loading. Photoelasticity, then, will unquestionably make of barodynamics a greatly more useful tool than it could ever be alone and may even make barodynamics useful in cases where it could not otherwise be so. He has also used the photoelastic method on statically loaded models, but it seems that it will be most useful in connection with the centrifugally loaded models of barodynamics because of the truer simulation of the force of gravity.

All of the foregoing can be summed up by saying that I am willing to grant the validity and potential usefulness of photoelasticity without any arguments. I am, therefore, going to assume that most mining men will not have any hesitancy about accepting photoelasticity if they can convince themselves that barodynamic methods are applicable to their problems.

The first reaction of many mining men to barodynamics has no doubt been that the method will probably work very well for the simple and uniform set of conditions assumed for the laboratory experiments but will fail to yield information of value when applied to the variable and complicated conditions found in the field. That might have been my reaction also if I had not been fortunate enough to have some field experience which closely agreed with the laboratory findings. At the time when I read his paper on longwall mining a couple of years ago I had just completed the daily supervision of a considerable amount of longwall mining covering a period of three and one-half years and involving more than 6,000 ft. of longwall retreat and the extraction of about 300,000 tons of anthracite coal. For roof support steel props were used which allowed the strata over the mined out area to fall without hindrance, in this respect resembling the condition assumed in his laboratory experiments.

I have already stated that my experience in the field agreed closely with experimental results and will now add that the study of the experimental findings enlightened me as to the significance of some of the phenomena I had observed in the field. This was true even though the conditions in the field were much more complicated and variable than the conditions assumed for the laboratory experiments. I therefore have a strongly grounded belief in the possibility of using barodynamics to aid in solving certain mining problems.

I would now like to subject these new methods to a more systematic and rigorous examination by asking the following pertinent questions about them.

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* Mine Foreman, Hudson Coal Co.

1. Are the methods theoretically sound?
2. Are these methods the best available for obtaining certain desired information?
3. Can field conditions be duplicated in the laboratory closely enough to permit use of experimental results for predicting results in the field?
4. What are the problems which these methods might aid us to solve?

Question No. 1 has been pretty well answered by Professor Bucky and his co-authors with regard to photoelasticity in the present paper and with regard to barodynamics in earlier papers. Personally, I could find no flaw in the theories and reasoning presented.

Perhaps it would be well at this point to emphasize that the foundation of the barodynamic method is the method of loading the models. Static loading of scalar models fails to truly simulate the force of gravity. By the use of centrifugal loading he overcomes this defect and is thereby enabled to obtain results which closely agree with field phenomena.

Now, as to question No. 2, "Are these methods the best available for obtaining certain desired information?" This question can be easily answered by stating that when principles of similitude are adhered to, scalar models represent the most satisfactory and economical way of obtaining information about processes which if they had to be investigated on full size models would be unduly tedious, cumbersome and expensive. The foregoing statement is not to be taken as a criticism of certain experiments now being conducted on pillars in place in the mines, nor of recent studies of roof behavior by means of subsidence-recording devices. Such investigations should be of equal value and importance with barodynamic investigations and it is to be hoped that in the near future results of these various lines of attack will be correlated with one another to the end that each may serve to enhance the value of the others.

In answering the question with respect to photoelasticity, we can state that it is the only known means of determining the actual location, direction and amount of internal stresses in a structure.

Next, as to question No. 3, "Can field conditions be duplicated in the laboratory closely enough to permit the use of experimental results for predicting results in the field?" My own experience has been too limited to give an unqualified answer to this question. With regard to photoelasticity I will venture the opinion that if the question can be answered in the affirmative for the barodynamic method it can also be answered in the affirmative for photoelasticity.

Just for what it might be worth in providing an answer to the question, I would like to compare briefly at this time my observations of longwall work in the field with the laboratory findings.

1. Professor Bucky concluded that the minimum length of longwall face should be preferably at least five times, and in no case less than one and one-half times, the span at which the overweight will fail as a beam. I am not able to say at what length of span the overweight failed in the longwall work which I observed in the field, but I can state that under the conditions that prevailed a longwall 360 ft. long worked very successfully, while shorter lengths of face gave considerable trouble. In one case, owing to convergence of two gangways, a longwall of decreasing length was worked, and as the length decreased, trouble began to be experienced because the roof would not break and therefore threw excessive weight on the steel props and the longwall face. This condition became so bad that when the length of face decreased to 120 ft., the longwall method of working had to be abandoned. This agrees with the conclusion that for a given condition there is a minimum length of longwall that can be satisfactorily worked. My observations also lead me to believe that there is a relationship between the span at which the overweight fails as a beam and the minimum suitable length of longwall face. However, because I have had no opportunity

to make precise observations since I read Professor Bucky's paper, I cannot say how closely his numerical ratios agree with an actual case in the field.

2. A cyclical variation in roof pressure was characteristic of the longwall work in the field. This conforms with his observation of periodic failure of the overweight and his photographs of stressed models make the reasons for this cycle clear to us instead of a matter of conjecture as it formerly was. In practice we found that there was a certain phase of the cycle when a longwall face could be safely stopped and allowed to stand for an indefinite period of time, which also agreed with Professor Bucky's conclusions.

3. His conclusion that desirable longwall conditions are a thin strong underweight and a thick strong overweight also agrees with my field observations.

Since this is properly a discussion of photoelasticity, and only incidentally of barodynamics, I will not go into further detail as to my longwall observations at this time.

Finally, as to question No. 4, "What are the problems that these methods might aid us to solve?" I am sure that most of you can think of many such problems and I only want to mention a few that have come to my mind.

1. In localities where "bumps" occur due to pressure from overburden, the hazard might possibly be reduced by studying occurrence of stresses with a view to changing method of mining and design of pillars so that stresses will not occur where they favor the occurrence of "bumps."

2. Study of roof and pillar stresses when robbing pillars under strong roof where "squeezing" causes loss and expense.

3. Study of stresses in robbing pillars which have a high ratio of height to width in order to devise safer methods of mining.

4. Effect of stratification and cleat in pillars on the stresses produced by robbing.

5. Effect of pillar robbing in an overlying bed on the roof and pillars of an underlying bed and particularly whether methods of pillar robbing can be devised which will minimize the bad effects of noncolumnization of pillars.

6. Planning of development and method of mining for a new mine.

It seems to me that photoelasticity and barodynamics come off very well from the foregoing examination. I therefore feel justified in summarizing my conclusions with regard to these methods as follows:

1. Photoelasticity will unquestionably be an integral and valuable part of the method of observation employed, if the barodynamic method comes into practical use.

2. Study of available data indicates that barodynamics combined with photoelasticity offers a most promising means for aiding in the solution of numerous difficult mining problems.

C. T. Holland,* Lookout, W. Va.—The use of models and the application of the principles of similitude for studying action of mine structures in the laboratory offer about the only means of investigation in which the various factors being studied are under the control of the investigator within reasonable limits. In addition to this advantage, men and valuable equipment are not subjected to any hazard as they may be in a similar experiment made in the mine. Furthermore, the cost of the experiment itself is low as compared to a similar experiment in an operating mine.

The present study illustrates one of the many problems that may be investigated by the use of models. It seems, however, to be important to remember that the results obtained in this study apply principally to a comparatively shallow roof, a thick bed that is structurally as strong as the roof, a strong roof and particularly a roof without joints in the rocks composing it. These factors probably explain the remarkable

* Superintendent of Mines, Lookout Smokeless Coal Co.

uplift or negative subsidence observed, which reached a maximum amplitude of 0.865 ft. or 10.4 in. In so far as is known to this writer this is about five times the maximum of about 2 in. observed in the field.[10] This would suggest that in some places the uplift, which occurs in instances, could cause considerable damage.

The observation of the authors upon the arch action is particularly apt. Possibly in the past writers upon roof action have depended too much upon beam action for their explanation of the phenomena observed. This seems particularly true in regard to weak strata, weak either because of weak structural material or because of a well developed set of joints in a strong structural material. In the present case, however, observation of the illustration seems to indicate that cantilever beam action occurred and played a considerable role in the roof action. For the sand filling, however, it seems to be indicated that the various roof members, except possibly E, acted as beams uniformly loaded, partly restrained at the end supports and resting upon a yielding support between the end supports. This view is based upon the facts that no cracks are to be observed in the roof strata and the authors do not mention the formation of cracks.

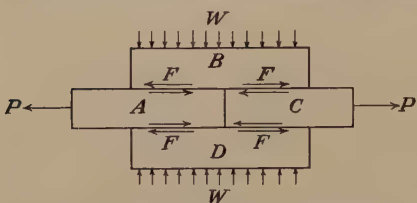

FIG. 29.—ILLUSTRATION OF TRANSFER OF TENSION ACROSS A JOINT PLANE.

The influence of joints on roof action is not well understood. That roofs composed of strata having well developed joint systems can and regularly do span considerable areas is well known.* A careful consideration of a roof of this nature seems to indicate that such a roof cannot transmit tension across the joints and therefore cannot act as a beam. It may act, however, as an arch. These statements apply to the strata over the mined-out areas. Over the solid supports, however, because of the weight of the overlying strata, the solid subjacent support, friction between the strata and the fact that the blocks cut out in one stratum by the joint system do not necessarily coincide with the dimension of the blocks cut out in a superjacent and subjacent stratum, tension may be transmitted across the joint planes. Perhaps the meaning may be made clear by Fig. 29. An arch transfers a considerable load on the ore body forming the abutments of the arch, which causes the ore body to deflect and allows the roof to deflect. This causes the roof to elongate slightly, thereby throwing a tension in the roof. Considering a section of it as shown above, the force P acting on a block tends to slide the block; but before the block can slide the frictional force F resisting motion must be overcome. If the force F is sufficient the block A cannot move and the force P is transmitted to blocks B and D, which in turn transmit the force to block C. Through this kind of action the roof overlying the solid ore body, or ore body split only by narrow headings, may act in a manner that, while not beam action, is certainly analogous to beam action and can and may cause action similar to beam action. The tension transmitted in any particular stratum in this manner would depend upon the strength in tension of the material forming the stratum, the depth of the stratum from the surface, the distance apart the joint planes are spaced, the coefficient of friction and in some cases the mechanical bond between the stratum under consideration and the stratum above and below.

[10] W. Thorneycroft: Effects on Buildings of Ground Movement and Subsidence Caused by Longwall Mining. *Trans.* A.I.M.E. (1931) **94,** 52–64.

* Bucky [A.I.M.E. *Contribution* 4 (1933)] has shown that a mine roof composed of one layer is not greatly weakened (less than 15 per cent) if the cracks are approximately vertical, the side of all cracks are in juxtaposition, and the mine roof is securely fixed at its supports.

TABLE 3.—*Observations on Pillar Compression in Coal Mines*

Observer	Distance Ahead of Face Coal Began to Be Deformed, Ft.	Type of Mining	Thickness of Cover, Ft.	Maximum Compression at Working Face, In.	Thickness over Which Convergence was Measured		Amount of Re-elevation, In.	Distance from Face at Which Re-elevation Occurred, Ft.	Amount of Convergence When Re-elevation Occurred, In.
					Ft.	In.			
Briggs[a]	9 to 20	Longwall	135	0.277[j]		24	None observed		
Greenwald and Rice[b]	140[g]	Room and pillar[i]	297–350	1.3[l]	5.5		None observed		
Winstanley[c]	70	Longwall	1250	2½	4		0.01 and 0.015	15–25	0.01 –0.02
McCall[d]	458[h]	Retreating longwall	In excess of 2000	26.9[k]	9		0.0625–0.3800	233–214 284–276	1.7500–2.625
Holland[e]	82–215	Room and pillar[i]	112–173	1.55[j]		42	0.0013–0.0049		0.003 –0.009
Bryson[f]	Only at face	Room and pillar	1200–2000	Not stated[m]		Averaged 46	None observed		

[a] H. Briggs and W. Ferguson: Investigation of Mining Subsidence at Barbauchlaw Mine, West Lothian. *Trans.* Inst. Min. Engrs. (1933) **85,** 315.
[b] H. P. Greenwald and G. S. Rice: Studies of Roof Movement in Coal Mines. U. S. Bur. Mines *R.I.* 3355, 21.
[c] A. Winstanley: Longwall Roof Control. *Coll. Guard.* (1931) **142,** 746, 2224.
[d] T. L. McCall: Further Notes on Bumps in No. 2 Mine, Springhill, Nova Scotia. *Trans.* A.I.M.E. (1934) **108,** 61.
[e] C. T. Holland: Pillar Deformation in a Bituminous Coal Mine. *Trans.* A.I.M.E. (1938) **130,** 345.
[f] J. F. Bryson: Method of Eliminating Coal Bumps or Minimizing Their Effects. *Trans.* A.I.M.E. (1936) **119,** 49–50.
[g] Read from curves. Authors state important convergence within 70 ft. of working face.
[h] Face 31 ft. distant when station was destroyed.
[i] Essentially a retreating longwall.
[j] Deformation measured to 0.001 inch.
[k] Deformation measured to 0.0001 inch.
[l] Deformation measured to 0.01 "rounded off" to 0.05.
[m] Accuracy of measurement not definitely stated but from description probably was not closer than 1/16 inch.

That the ore body adjacent to a longwall face or room-and-pillar retreat line is subject to compression has been demonstrated not only by my researches but by others as well. For a convenient reference some of the observations are reproduced briefly in Table 3. It is interesting to note that in six independent investigations five report deformation of the coal bed in advance of the working face. In this one exception the evidence is not conclusive that compression ahead of the working face did not occur, because the method of measuring compression probably would not indicate convergences of less than 0.05 or 0.06 in. It is also interesting that two observers, in addition to myself, noted a re-elevation or decompression ahead of the working face. As Bucky and Taborelli stated, this indicates that in some cases at least the load on the pillars near the working face of longwall workings and the retreat line of room-and-pillar works is variable. It is also important to keep in mind that three of the observers do not record this phenomenon: Briggs and Ferguson, who conducted a very complete investigation, specifically state that it was not observed by them (p. 314 of ref. on p. 257). This has been confirmed by correspondence.

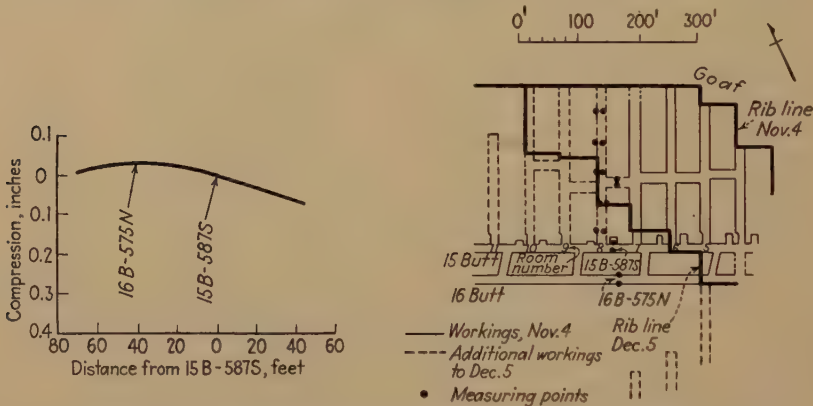

FIG. 30.—DIVERGENCE OBSERVED BY RICE AND GREENWALD.
Sketch of mine working traced from U. S. Bur. Mines *R. I.* 3355, Fig. 7.

In addition to the data quoted above, Rice and Greenwald observed a very interesting and perhaps a very important kind of roof action. They describe this action (p. 20 of ref. on p. 257) as follows: "Apparently the roof over the chain pillars between 15 and 16 butts was tilted towards the goaf, for on November 16 there was as yet no measurable movement at 15B-587S but 0.03 inch *divergence* at 16 B-575 N. At the end of the series convergence was 0.09 inch at the former point and 0.02 inch at the latter. More detailed information on the behavior of the roof adjacent to chain pillar comes from the third series of measurements." This measurement was made in 5.5 ft. of coal overlain by 9.8 ft. of weak bands of coal and slate, then with 49.2 ft. of a strong sandstone or shale bed, which would act as a monolith and with few joint planes (p. 13 of ref. *b*, on p. 257). The action described is visualized by myself as shown in Fig. 30. It seems clear that in this case beam action is occurring to produce the divergence described, and clearly part of the load on the coal produced by the overlying strata is relieved by the divergence observed.

Whether or not the divergence described above occurs regularly in advance of a longwall face or room-and-pillar retreat line cannot be stated definitely from field records. Rice and Greenwald observed the occurrence only once at Montour. Although looking for such a phenomenon, I did not detect any divergence that could be called significant in view of the accuracy of my measurements (p. 348 of ref. on

p. 257). Briggs and Ferguson also failed to note such a divergence. Such an occurrence is not mentioned by McCall or Winstanley. Clearly the investigators of roof and pillar action have not noted a divergence or negative subsidence. This may be accounted for on the ground that the measurements are not sensitive enough to indicate divergence or that it does not occur in every case. It is certainly a phenomenon that should be more carefully investigated because it sheds a great deal of light on roof action.

R. D. Hall,* New York, N. Y.—Studies into the application of photoelasticity to roof-control problems are to be welcomed, even though we are not sure that semielastic bodies like coal and rock will behave like plastics. Caution is necessary but, that granted, one cannot but feel that much can be learned from Professor Bucky's experiments, which afford confirmatory evidence to a growing belief that "brows" near the roof are a source of weakness, not of strength, and are to be condemned. A pillar with a trapezoidal cross section in all directions with the smaller dimensions at the roof would be preferable to a pillar with vertical sides or with a "mushroom" top, but such a pillar would be difficult to construct.

Professor Bucky has stated that if a pillar is split by driving an opening through it so that two small fenders are left, one on each side of the split, the purpose of the undisturbed pillar still will be served, provided a little coal is left near the roof to take care of the heavy stresses that his photoelastic studies have revealed as confined to that part of the pillar. One wonders whether it is necessary to leave the roof coal in place and whether, when it is removed, the strains will arrange themselves in similar manner in the roof and will be as inconsiderable at the lower edge of the rock as Professor Bucky has shown them to be at the lower edge of the coal slab. Here, the lesser elasticity of the rock may prevent a similar stress distribution. Of course, in a cut-through near the end of the pillar, one fender is small and the other "fender," if "fender" it may be termed, may be 50 ft. thick. In that case, is there any difference in the stress diagram?

T. Ertl† and M. A. Smith,‡ Kimberley, Nevada.—By laboratory experiment Sinclair and Bucky are reaching conclusions that observing operators have long suspected. Other researchers, Terzaghi, Hogentogler, Zisman, Carstarphen, and others, are arriving at conclusions that will benefit the mining industry.

Little is known about the action of pillars above the mine openings. A pillar may help support the mine opening by absorbing a portion of the weight, or by directing the weight to another area where it will do no damage. Paradoxically, a pillar may operate as a gathering point for the weight above and may concentrate the load in a small area.

In a Western porphyry copper mine, thin, horizontal pillars are left above the workings to try to protect them from the weight of the overlying caving ore. An attempt is made to destroy all pillars within the ore because experience has shown that these pillars often act as a fulcrum for a weighty mass, and that the pillars collapse the openings in the bottom rock. Consequently, we are most interested in the experiments that deal with stress distribution and direction in the openings under the pillars.

That stresses vary greatly in these bottom rock openings, both in intensity and direction, has often been observed. We have noted many pressures breaking 12 by 12-in. posts in a period of a few days, yet only a few feet away lagging of 12 sq. in. cross section is left intact. Bottom pressures occasionally break 12 by 18-in. sills

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* Engineering Editor, *Coal Age.*

† Planning Engineer, Consolidated Copper Mines Corporation.

‡ Consolidated Copper Mines Corporation.

while not injuring 8 by 8-in. foot braces within 3 ft. of the breaking sills. Further proofs of pressures varying greatly within a short distance could be cited.

The presence of a pillar in the ore can always be detected—but generally too late—for the pillar may crush the workings before it can be reached and removed. There are many examples in which a drift stands well with little maintenance while a neighboring drift 30 ft. away is crushed completely, so that it must be redriven. Also, a portion of a small drift stands in as good condition as when driven yet in another portion 40 ft. away caps, posts and sills break with alarming rapidity.

That stresses vary in direction has also been noticed. Often timbers from the opposite side of a drift are acted upon by compressive forces until they meet in the center of the drift. Because of varying directional stresses, timbers on the sides of drifts, such as finger raises, are raised or lowered many feet in respect to their original position. In a drift driven on line for approximately 100 ft., we have observed formation of a curve by pressure pushing the center of the drift 5 ft. out of line while leaving the ends in place.

We have seen set 2 of a drift push away from set 1 and toward set 3, causing the collar braces between set 1 and 2 to fall out, and causing the collar braces between sets 2 and 3 to break. Many times we have seen set 2 rise 12 in. or more from its original position while sets 1 and 3 remained stationary.

Does not all this evidence bear out the conclusions reached by laboratory tests?

After observing the fine results of the preliminary studies of rock stresses achieved by research during the past few years, we think that money spent by the mining industry encouraging this research will pay large dividends. Maintenance of extraction openings in the mines of the world is costing the mining industry many thousands of dollars. Much of this money will be saved when the laboratory is used to its fullest extent in the study of barodynamics.

I. Hartmann* and H. P. Greenwald,† Pittsburgh, Pa.—Within recent years photoelasticity has become an important tool for the structural and mechanical engineer, particularly in designing complicated structures and parts in which high stress concentrations are likely to cause failure. Mathematical treatment of many stress-distribution problems is very laborious, if not impossible; in such instances the information obtained from photoelastic stress patterns of small-scale models is often of great value. Application of the method to the study of mine problems described in this paper is noted with great interest. The general purpose of the study is highly desirable, and the authors are to be congratulated for making a start in this direction.

It is important that the limitations of the photoelastic method and its practical applications be clearly realized. The general theory of stress analysis is based on certain definite assumptions, which include the homogeneity of the stressed solid body. Photoelastic analysis is based on similar assumptions, and this fact must not be forgotten in interpreting photoelastic patterns and drawing conclusions for large-scale structures therefrom.

The stress patterns obtained with homogeneous Bakelite models used in this work represent conditions in homogeneous large pillars of similar shape and under like loading conditions. They do not represent the stress distribution in actual mine pillars of coal or stratified rocks that are not homogeneous or isotropic but contain cleats, slips, bedding planes, and inclusions of foreign materials. All these factors influence the stress distribution, the strength, and the mode of failure of mine pillars in a manner

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that cannot be duplicated in simple Bakelite sheet models. This difficulty has been in effect one of the greatest hindrances to all theoretical and laboratory studies of rock strata and is one reason why no structural calculations, barodynamic tests or other model experiments have so far yielded satisfactory results for determination of the required dimensions and shapes of mine structures, except upon the basis of greatly simplified assumptions.

That bedding planes and inclusions of foreign materials having strength, hardness, and elastic properties different from those of the principal pillar material affect the stress distribution in pillars is self-evident. That cleats and other planes of weakness in coal influence the strength and manner of failure was found in tests by the Bureau of Mines and other investigators. In a recent series of compression tests of mine pillars in the Pittsburgh coal bed conducted by the Bureau,[11] it was determined that square pillars comprising one-half the height of the bed failed under a pressure several hundred pounds per square inch greater than full-height pillars of geometrically similar shape loaded in exactly the same way. This resulted partly from difference in effect of the cleats on stress distribution and strength in the two instances.

In coal mines the load supported by any given pillar depends on the nature of and the stress distribution in the roof strata, the nature of the immediate roof, the dimensions of the pillar, its relation to surrounding pillars, and other factors. Some pillars are loaded almost uniformly across the entire top section; in others the outer edges at the top are loaded most; while in still others the edges are loaded less than the central part. Consequently, in some instances pillar failure is determined by the maximum normal stresses and in others by the maximum shear stresses. The type of loading used in the model tests described in the paper, in which the pressure was concentrated at the upper corners or edges, is approached in coal-mine pillars over which the roof bends as a beam or plate. It is most unlikely, however, that the immediate roof over any actual pillar is completely lifted off the latter at the center, as shown in the tests.

The stress patterns in the Bakelite pillar models (Figs. 5 and 7, pp. 227 and 229) indicate high shear stress concentrations at the upper corners. It is not safe to conclude from this that these stresses govern the failure of actual pillars. What happens frequently in coal-mine pillars is that the upper edges spall off; but the pillar as a whole remains virtually as strong as before, and, in fact, the stresses may then be distributed more favorably than before the spalling, as shown in Fig. 8 (p. 231). Deliberate cutting away of the edges may result in weakening a pillar when there are marked cleats near the outer boundaries or when the floor is of a yielding character. Under both of these conditions the outer layer of coal is to all intents and purposes removed from the pillar. When the immediate roof adheres strongly to the coal, cutting away the edges or corners might result in much higher stress concentrations at the reentrant angles. Such conditions cannot be predicted from the stress patterns. Figs. 5 and 7 also show that the stresses in the central part of the vertical edges are comparatively small. This is quite reasonable; nevertheless, experience in coal mines and the Bureau of Mines tests mentioned above indicate that failure of many coal pillars begins by separation or spalling along the cleats of these outer sections, which are comparatively stress-free; and as the failure progresses more and more coal is loosened from the exterior of the pillar until the section is greatly reduced in area and the pillar is crushed completely.

In discussing the effects of openings in pillars, the authors conclude that a pillar of the shape in Fig. 10*c* (p. 232) is able to support the same load as that in Fig. 10*a*.

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[11] H. P. Greenwald, H. C. Howarth and I. Hartmann: Experiments on Strength of Small Pillars of Coal in the Pittsburgh Bed. U. S. Bur. Mines *Tech. Paper* 605 (1939). 22 pp.

They do not consider the high stress concentrations near the upper corners of the opening (this concentration could be reduced greatly if the corners were rounded off); when these are tensile stresses the pillar may fail from this cause, otherwise it probably will shear along the diagonal planes connecting the upper inner and outer corners. Furthermore, in actual mine pillars the legs of pillar 10*c* might contain planes of weakness that would cause failure under smaller load than is required for pillar 10*a*.

A point not considered in the paper is the effect of floor conditions on the stress distribution of mine pillars; in some instances this is the chief factor governing pillar failure. Two of the pillars tested by the Bureau of Mines failed when the lateral flow of the yielding fire-clay floor dragged the pillars apart along vertical cleats; rupture started at the base of the pillar.

C. T. Holland,* Lookout, W. Va.—If an engineer attempts to apply a mathematical analysis to a problem involving stresses in mine roof and pillar structures, in all except the simplest cases, he runs into difficulties that so far generally have been insurmountable, because the factors entering the problems are so numerous, complex and variable, and in some cases unknown. If he enters a mine and attempts to measure the stresses, many problems, so far baffling, prevent him from obtaining the information wanted, other than the amount of strain occurring.[12] It must be conceded that the information on this kind of work is so meager that it is hard even to interpret the strains in a satisfactory manner.

Tests of models in a centrifuge have demonstrated that the actions of mine structures can be investigated in this manner to a considerable extent, although it seems to me that it will be extremely difficult to produce a model of a mine structure on a scale small enough to be used in a centrifuge and yet accurately represent the mine structure.[13] The investigation of models up to this time, however, has not revealed very much about the actual stresses to be encountered. So far only the actions of the models have been recorded.

Sinclair and Bucky have used a method of analysis that is new to the investigation of mine structures. The method of photoelasticity has yielded accurate and valuable results in other fields of engineering. The method of photoelasticity not only indicates where the maximum stresses occur in a model of a structure, but also supplies the means of calculating the value of these stresses. There is one disadvantage, however—that the values of stresses calculated will have to be found from models built of some transparent isotropic material rather than the material making up the roof, pillars, and floor of mines, which is, of course, not isotropic or transparent.

In other branches of engineering this has not been such a serious handicap, because, knowing rather completely the characteristics of the material of the model and of the prototype and the details of the construction, accurate models could be built and the relationship between the model stresses and strain and that in the prototype could be readily worked out. In mining, it seems that this ideal condition lacks so much of fulfillment that it will be rather hard to apply the results obtained from model study to the prototype. This is not due to theoretical difficulties, but to the practical difficulty that we are generally very much in doubt as to just what the model is to represent, and frequently not well informed concerning the physical properties of the material composing the prototype. As an example, consider Fig. 12 (p. 234), showing the model used to investigate the effect of arching in mine pillars. In actual mining, we are usually rather uncertain about the amount of bond between the pillar and the roof. In addition to this, we are even more uncertain about the bond between

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[12] C. T. Holland: *Trans.* A.I.M.E. (1938) **130**, 356.

[13] C. T. Holland: *Trans.* A.I.M.E. (1934) **109**, 48.

the various strata composing the roof. This means that while we know the thickness of the cover we do not know accurately the thickness of the beam that spans the passageway and rests on the pillars. From a study of the stress patterns shown by the authors, the stresses caused are shear stresses set up by the deflections of the beam forming the roof. The deflections of beams and, therefore, the shear stresses at the edges of pillars will depend upon: the length of span, the elastic constant of the material comprising the beam, the depth of the beam, and the amount of restraint (partly depending upon the mechanical bond between the pillar and the roof) and the depth of the beam (which will be doubtful because of lack of knowledge of the cohesion between the strata composing the roof) will raise real obstacles to the construction of the model.

The effect of arching the pillar, as Bucky and Sinclair show, is to increase the shear stress at the arch and to cause this portion of the pillar to fall off or spall. This may occur even when the pillars are not arched. In mining, this condition frequently exists. However, observations made in the field indicate that structural conditions may alter this. In the experiment described, the photograph shown in Fig. 12 seems to indicate a span of 8 to 10 times the thickness of the roof beam; in addition, a lead load was applied to the roof. These conditions are favorable to relatively large deflections. In coal beds with pronounced jointing, another condition hard to reproduce in models, I have noticed the pillars spalling and the failure was frequently at approximately the center of the seam. Apparently the failure was due more to the slabs of coal formed by the joints failing as columns rather than in shear.

It is a fact easily verified by observation that mine roofs frequently form arches. While these are not usually in the structure left on the pillar, they form part of the pillars after the arch forms. Evidently mine roofs frequently form an arch due to the nature of the material and the stresses acting. It seems to be a reasonable conclusion that an arch so formed is the best type of structural shape for this purpose. Were the material forming the pillars the same as that forming the roof (a condition frequently existing in underground quarries) it would seem to be a good idea to shape the opening to conform to the arch. This would be nothing more than arching the pillar. The principal objection is that we do not know the condition under which arches form in mining, or the factor influencing their shape. Considerable observation in coal mining seems to point to the fact that roof arches form best when the members composing the roof are thick members with local weaknesses. A roof composed of many thin bands, however, may arch if the bands are strong. Thin, weak bands, such as draw slate, have a tendency to shear along the edge of the pillar and are slow to arch.

There is another problem in which an investigation using the methods of photoelasticity and the centrifuge might yield some interesting information. Greenwald, Maize, Hartmann and Rice,[14] McCall,[15] Winstanley,[16] and Holland,[17] when investigating the action of pillars close to an advancing pillar line, or over the solid coal in longwall work, have apparently noticed that the roof first shows a subsidence reaching a distance in advance of the working face up to several hundred feet in room-and-pillar work, much less in longwall work. Then, as the working face or pillar line

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[14] Greenwald, Maize, Hartmann and Rice: U. S. Bur. Mines *R.I.* 3355, 20–28; *R.I.* 3452, Fig. 13.

[15] T. C. McCall: Further Notes on Bumps in No. 2 Mine, Springhill, Nova Scotia. *Trans.* A.I.M.E. (1934) **108,** 55.

[16] A. Winstanley: Longwall Roof Control. *Coll. Guard.* (1931) **142,** 747, 2224.

[17] C. T. Holland: Pillar Deformation in a Bituminous Coal Mine. *Trans.* A.I.M.E. (1938) **130,** 348.

approaches closer to the point of observation, an uplift is noticed at times. Fig. 31 has been constructed to make clear the occurrence described.

A study of the possible causes suggests that the uplift may be caused: (1) by the top acting as a beam, the coal pillar between the point of uplift and the coal face acting as a fulcrum and the uplift being actually an elevation of the entire overburden comprising the roof of the mine; (2) by a decrease in the load on the pillars caused by major readjustment of the material lying over the mined-out area; this would apply regardless of whether the load thrown on the coal pillars came from beam action or arch action. While these effects have been observed, it seems to be extremely difficult and costly to interpret them without resorting to some method such as those illustrated by Sinclair and Bucky. The strains are so small that using a model in a centrifuge and attempting to measure the strain itself would seem to be almost impossible. But possibly the stresses set up would be indicated by use of a model in a centrifuge on which it was possible to apply the methods of photoelasticity. These stresses, if properly investigated, may lead to some valuable information in roof action and a number of related phenomena such as bumps and rocks bursts.

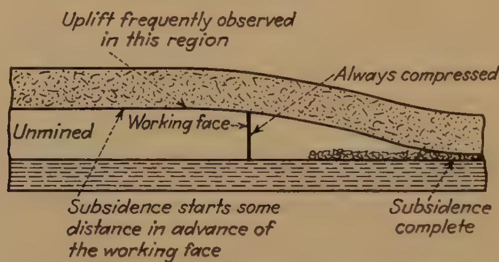

Fig. 31.—Roof action in coal mines.

The application Bucky and Sinclair have made in investigating the stresses set up in the walls in a tunnel probably have several important applications. The locations of passageways in a lower seam relative to pillars in an overlying seam would seem to be one. Apparently at times there are some stress effects that show up in this connection that are not understood. In a case of this kind, with numerous pillars entering the problem, a mathematical solution based on the laws of mechanics would probably be impossible while an investigation using the methods illustrated by Sinclair and Bucky possibly would give some valuable results.

The results as obtained by photoelastic and barodynamic methods agree very well with those calculated mathematically from the theory of elasticity. It would seem that the results obtained by model experiment would be more reliable than those obtained by mathematical methods, because the results obtained by mathematical methods are based upon the assumption that pressures are hydrostatic at depths and that the pillars rest upon an infinite plane. We know, of course, that neither condition is completely realized.

The authors are to be congratulated upon this paper for the results obtained, for the interesting methods of investigation they have introduced and for the ingenuity they have shown in applying the optical methods of investigation to a model in a centrifuge rotating at a high rate of speed.

P. B. Bucky (authors' reply).—Since part of the discussions include comments on two papers, it was considered advisable to group all together. These comments, coming from anthracite, bituminous, research, editorial, and metal-mining men, present the views of a fair cross section of the mining profession.

Mr. Carl Peterson sums up well the reactions from men in the field to this type of investigation. He also compares the results of his field experience in anthracite longwall mining with the laboratory results and conclusions presented in the paper beginning on page 211. In addition, he provides evidence that the results in the laboratory obtained over a period of a few hours or days at the most are the equivalent of what takes place in the field over a much longer period of time. This statement and its implications will be made clearer in the following reply to Mr. Holland's first discussion, on negative subsidence.

Mr. Holland presents field data relating to negative subsidence or divergence of the ore body ahead of the working face. The data show that negative subsidence has been observed, is of questionable presence in some cases, and is not present in others.

The authors, however, call attention to paragraph 3, page 222, where it is stated that "The cause of the rise ahead of the working face is the inverted arch action of a particularly strong thick bed in the geological series overlying the ore body. One would therefore expect little rise where the geologic beds were thin or weak in shear or compression, or if support was so placed as to materially affect the curvature of the inverted arch." To the previous statements may be added the fact that the Pittsburgh bed, where the Greenwald-Rice observations were made, is overlain by a very strong sandstone bed. Brief study of some of the sections overlying the coal beds in England show comparatively great depths of thin layers of weak geologic material overlying the coal beds, a condition not conducive to a negative rise. In addition, the amount of cover may be such, i.e., the restraint over the coal bed so great, that the effect of the bending of a relatively thick bed may be very little. Therefore the English observers do not report it.

To the question as to whether the divergence occurs regularly in advance of a longwall face or room-and-pillar retreat line, the evidence from barodynamic tests is as follows: (1) When the retreat first starts the divergence is negligible; (2) as the retreat continues the divergence effect is increased; (3) a retreat distance will then occur during which divergence may not be noted underground but the pressure on the coal face is reduced. This occurs when the thick beam has failed so that it rests on the bottom and is exerting an upward force on the unbroken portion of itself and consequently the beds above it (Fig. 1, 317 ft., page 212). (4) A retreat distance is then encountered when face pressure and divergence are noted again and 2 and 3 repeat themselves.

The negative subsidence effect may be demonstrated by placing a ruler on a table and allowing one end to project. If the end of the ruler is held down with one hand and the load applied on the projecting end with the other, the portion on the table between the hand and the table edge will be seen to rise.

The preceding illustrates the possibility of obtaining a variety of evidence regarding negative rise in a mine, and that only, by a complete understanding of the factors involved, and data taken over a long period of time in the field can results be obtained that would compare with that obtained in a short time in the laboratory.

That this tendency to divergence will be present if the roof has a series of approximately vertical cracks whose sides are in juxtaposition is the opinion of the authors. That Holland's reasoning for the transmission of tensile stress in a roof portion over a pillar is true, but it must be remembered that these forces acting on the roof on both sides of the pillar can only transmit compressive forces to the roof over the open span and that tensile forces cannot act across a crack. When it is remembered that most rocks (p. 258 of ref. 20) have crushing strengths from 10 to 20 times their tensile strengths it is obvious that most loads must be taken care of by compressive stresses.

Messrs. Ertl and Smith present evidence obtained at a block-caving property tending to substantiate the results showing the large changes in stresses and therefore forces which act on the supports of mine openings. Block caving, a process whereby the ore is undercut and allowed to cave by itself, while at present not completely understood, is being successfully accomplished. It illustrates well the fact that some mining procedures and methods are in advance of our complete understanding of them. This problem is now being attacked in the laboratory with the hope that a better understanding will result in some contribution to the excellent results now being obtained.

It is the authors' opinion that many of the problems mentioned by Messrs. Hartmann and Greenwald and Mr. Holland can be solved, since they feel that the limits set by the assumptions of homogeneity and isotropy are not as narrow as defined by Messrs. Hartmann and Greenwald. All materials are more or less inhomogeneous and anisotropic, but in spite of that fact results indispensable to all branches of engineering are obtained from calculations based on the above assumptions.

Furthermore, it appears possible to construct a model of two or more different photoelastic materials cemented together. This would provide observations of the stress distribution across the boundary between two materials of different elastic constants, as well as the effect of different kinds and amount of cement. The cement would be analogous to an included layer of foreign material or the binding between bedding planes, etc. Solakian[18] investigated several mechanical and civil engineering problems by means of models constructed of several pieces of the same material cemented together.

Similarly, models of other structures suggested by Hartmann and Greenwald have been or could be constructed. Preliminary tests of a roof-and-pillar model, made all in one piece, have been made in this laboratory. A better model of a roof that adheres strongly to the coal could be made by cementing two different materials together. No difficulty should be experienced in constructing and testing a model of a pillar supported by a yielding floor. The spreading effect on the pillar of the lateral flow of the yielding floor material should show up very clearly in the stress pattern in the pillar and recently has been observed in barodynamic tests. The effect on the floor on approximately this type of loading has also been observed but not reported by the authors. It is not very different from that shown in Fig. 19. Photographs of two such types of loading are given by Nadai.[19]

The inhomogeneity caused by stratification of rocks results in anisotropy, the rocks being weaker in one direction than in another. This, however, does not appear to vitiate the results of the photoelastic method. The general effect of cleats in coal, for example, is to decrease the crushing strength of coal, but it does not seem evident to the authors that the stress distribution before failure will be altered by the presence of cleats. The experiments on coal cited by Hartmann and Greenwald do not appear conclusive, since they found a 30 per cent variation in crushing strength among geometrically similar pillars.

Such widely different materials as cast iron, sandstone and concrete increase indefinitely in strength as the height of column is decreased.[20] Photoelastic compression measurements made by Coker and Filon[21] indicate that this effect is due to the suppression of lateral expansion in short columns, causing a great difference in

[18] A. G. Solakian: Photoelastic Models with Cemented Elements. *Photoelastic Jnl.* (Jan. 1938); *Jnl.* Amer. Welding Soc. (Feb. 1934) 24; *Trans.* Amer. Soc. Civil Engrs. (1936) 940.

[19] A. Nadai: Plasticity, 248–249.

[20] Johnson: Materials of Construction, Ed. 7.

[21] E. J. Coker and L. N. G. Filon: Photoelasticity, 583.

stress distribution. This is the case when the columns are compressed between blocks of stronger material, and results in overload of the outside of the columns. When the pressure blocks are made of more yielding material, the inside of the columns becomes overloaded. The tests by Hartmann and Greenwald show a continual increase in pillar strength with decrease in percentage of lateral expansion at a constant pressure.

From their experience with the photoelastic-barodynamic method, the authors feel that this method can be used to determine the amount of the effect on the stress distribution caused by variations in the physical and mechanical properties of the materials. These effects are known to depend on loading and other conditions. The shearing strength of rocks was found by Bauschinger[18] to be much less when measured parallel to the bedding plane than when perpendicular to this plane. On the other hand, the crushing strength of a great variety of rocks was found by Griffiths[22] to be the same to within 10 per cent in all directions perpendicular or parallel to the bedding plane. This is to be expected but is mentioned to show that anisotropy even when large is not always an important factor.

Large inhomogeneities may be very important factors. In a pillar composed of more than one material, failure would start wherever the shear stress exceeded the shearing strength of the weakest material. This might or might not occur at the upper corners, depending upon the type of loading and the relative strengths of the materials.

Rounding the corners of the opening in the pillar of Fig. 10 would reduce the stress concentration at these points. This procedure was not adopted, since the stress observed by counting the fringes in these corners was always less than at the upper corners of the pillars. Fig. 15 illustrates a situation where rounding the corners causes an increase in stress. On the other hand, the type of arching mentioned below, where the radius of curvature is very large, reduces the stress concentration.

It should be pointed out that the results obtained from photoelastic models are strictly applicable only to prototypes stressed within the elastic limits. That is to say, the stress in a standing structure or in a part of a partly collapsed structure is determinable. In a structure that is to be collapsed, the point at which failure will start is determinable, but usually the progress of failure cannot be foretold, since the stress distribution is altered at the start of failure. However, since stratified materials frequently contain planes of weakness, it is known that shearing failure will take place along these planes. Advantage is taken of this fact in coal mining.

The discussion by Mr. Hall raises several interesting questions which have also occurred to the authors, and which they will endeavor to answer after the proper tests have been run. It is characteristic of most of the discussion that each has presented new problems of peculiar interest.

Failure tests on mine structures can be made by barodynamic tests on models made of field materials. For example, observations in this laboratory of roof uplift behind the fact (ref. 3 on p. 223 and p. 211, this volume) of the type observed by Holland and others in the field, led to the type of pillar loading used by the authors. The conditions that did or did not cause uplift were determined by this method. Arching of the type described by Holland was also observed in this laboratory (articles of ref. 1 and ref. 3 on p. 223, and p. 211 this volume) in barodynamic tests on sandstone.

The problem of the stratified roof suggested by Holland is one that should be investigated in detail. Barodynamic investigations of such a roof made from sandstone without bond have already been referred to (ref. 3). Photoelastic observations could also be made.

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[22] J. H. Griffiths: Iowa Eng. Expt. Sta. *Bull.* 131 (Mar. 1937).

The simple models used by the authors can, of course, yield useful results only for similar prototypes, similarly loaded. Other models of different structures or parts of structures under different loading conditions are being constructed and tested. It should also be borne in mind that study of an apparently complicated structure in all cases so far has resulted in resolving it into parts that may be studied by simple models. It must also be remembered that the basis for model tests is knowing your structure, and this can only be accomplished by borehole surveys and structural tests of the material comprised. Running a barodynamic test on mine-roof material without knowing the roof thickness or the material above it means nothing. Tests run show a maximum variation of the physical properties of geological material as being less than 20 per cent and approximating 5 per cent on the average. The variation in dimensional thickness, however, is sometimes rather great, as can be determined from geologic sections at most mines.

Whether because of our familiarity with the subject or the natural optimism of one engaged in this type of work, we feel that there are no insurmountable difficulties in the solution of the many problems presented in the discussion.

Interesting laboratory results are now available on block caving, the side pressures from loose materials, time effects in mining, and roof control, drawing from caved stopes, and the behavior of pillars of cleaty material when resting on a plastic or heaving bottom. As a matter of fact, plenty of work ahead is seen with results depending mainly upon assistance and finances available.

Launder and Table Washing of Fine Coal

By J. T. Crawford,* C. P. Proctor,† and J. A. Younkins,‡ Members A.I.M.E.

(New York Meeting, February 1940)

Coal-cleaning plants using the launder process generally wash the fine coal (minus 3/8 or minus 5/16-in.) separately in a plant consisting of washing launders or troughs placed one below another and sometimes supplemented by additional washing equipment such as tables and jigs. The coal and its associated impurities enter the feed end of the top launder, with a current of push water. As the material moves along the impurities, being heavier than coal, settle toward the bottom of the launder and are drawn off, with some coal, through "free discharge" boxes to the launder below. These boxes were given their name because their discharge falls freely; that is, it does not discharge into a water seal. Thus the impurities are progressively concentrated in the lower launders until box products are sufficiently free of coal to be discharged to refuse. The overflows of the upper launders go to clean coal while the overflow of the lowest launder, and perhaps that of the one immediately above, is usually recirculated to the plant feed. Thus the bulk of the clean coal is removed rapidly and the impurities are concentrated more slowly. The chief purpose of this paper is to present the cleaning data of such a plant under different operating conditions.

Operating Equipment

The washing units considered here are part of a complete plant cleaning minus 4-in. coal. The primary cleaned product is a high-quality gas and metallurgical coal and another portion of the output may be shipped as a steam coal. The units consist of two parallel Rheolaveur free-discharge units each containing five launders, together with three 7 by 14-ft. Plato tables. The launders are designated by the letters A-B-C-D-E (Fig. 1), launder A being the top and longest; E being the bottom and shortest. Launder details are given in Table 1. The launders are equipped with a series of free-discharge Rheolaveur boxes, which are integral parts of the launder (Figs. 2 and 3). Each box with-

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TABLE 1.—*Details of Fine-coal Launders*

Launder[a]	Rheo Boxes	Width, In.		1st Section[b]		2nd Section		3rd Section		4th Section		Distance between Boxes, Ft.[c]
		Top	Bottom	Length, Ft.	Pitch, In. per Ft.	Length, Ft.	Pitch, In. per Ft.	Length, Ft.	Pitch, In. per Ft.	Length, Ft.	Pitch, In. per Ft.	
A	7 double	28	23	20	2⅛	19	½	9	¼	30	⅛	4.75
	3 single	24	14									
B	9 single	22	14	8	2	20	½	12	¼	15	⅛	4.75
C	8 single	20	14	10	2	37	9/16	23	5/16			7
D	4 single	10		13	1 9/16	36	9/16					8
E	3 single	10		7	1⅞	19	9/16					8

[a] A, B, C launders are flared. D and E launders have vertical sides. Launder A is double for the first 53 ft. and single for the last 24 feet.

[b] So-called "classification section."

[c] Distance between boxes is only approximate. Boxes are closer together near feed end.

TABLE 2.—*Details of Free-discharge Rheo Boxes*

Launder	Box Number, Incl.	Type	Number of Slots	Box Size	Disposition of Box Product	Height of Barrage, In.	Remarks
A	1–3	6-FD-10	2	Double	To C launder	2	
A	4–7	6-FD-10	2	Double	To B launder	2	
A	8–11	3-FD-14	4	Single	To B launder	2	
B	1–4	4-FD-14	2	Single	To C launder	2	
B	5–9	3-FD-14	4	Single	To C launder	2	
C	1	4-FD-14	1	Single	To refuse	2	Vertical current
C	2–8	4-FD-14	2	Single	To D launder	2	
D	1–2	5-FD-10	1	Single	To refuse	2	Vertical current
D	3–4	5-FD-10	2	Single	To E launder	2	
E	1	5-FD-10	1	Single	To refuse	2	Vertical current
E	1	5-FD-10	1[a]	Single	To refuse	6	Fine box
E	1	5-FD-10	1[a]	Single	To refuse	6	Fine box

SIZE OF HOLES IN FREE-DISCHARGE RHEO BOX DISKS, DIAMETER IN INCHES

Hole Number	Disk No. 1	Disk No. 2	Disk No. 3	Disk No. 4
1	5/16	11/16	1 3/16	2 3/16
2	½	⅞	1 7/16	2 9/16
3	⅝	1	1¾	2 9/16
4	11/16	1 3/16	2	2 9/16
Blind	2⅜	2⅜	2⅜	2¾

[a] Covered with ¼-in. screen to permit passage of fines only, as these two boxes are used solely for removal of very fine slate, pyrite and calcite.

draws settled material through one or more slots in the bottom of the launder and discharges through an orifice into the launder below. Each

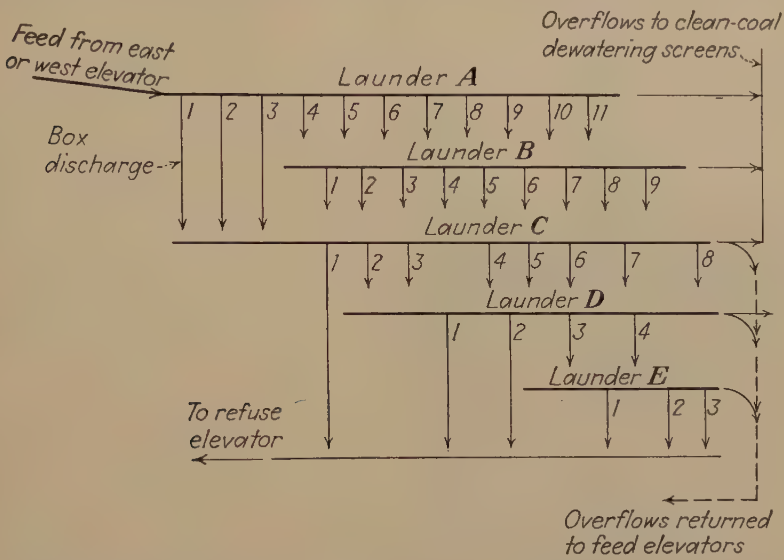

FIG. 1.—FIVE-LAUNDER FINE-COAL UNIT. EAST AND WEST UNITS IDENTICAL.

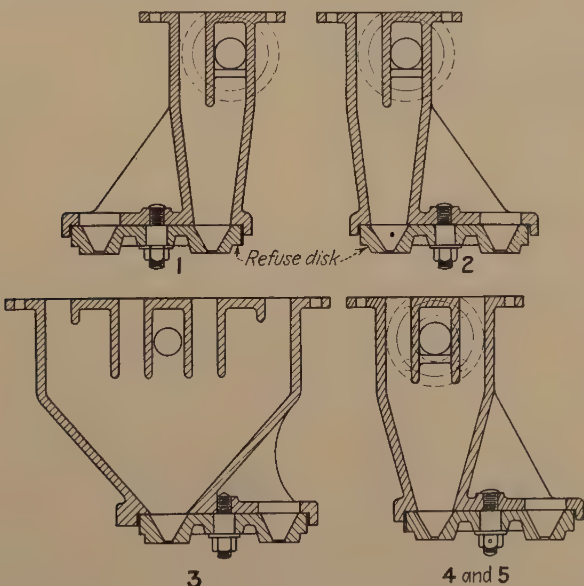

FIG. 2.—RHEOLAVEUR FREE-DISCHARGE BOXES, TYPES 1, 2, 3, 4 AND 5.

box has a movable disk, which has conical orifices (or holes) of varying diameter and one hole of the same diameter as the discharge of the box proper. By revolving the disk any one of the five holes provided may be

brought under the box, thus controlling the discharge. Four sizes of disks are available, with holes varying from $\frac{5}{16}$ to $2\frac{3}{4}$-in. diameter (Table 2). A more flexible control is obtained with a newer type of box, which has an easily replaceable bushing instead of the disk.

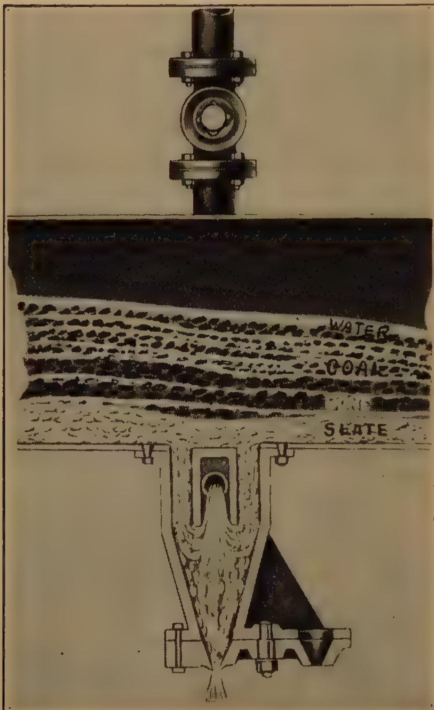

Fig. 3.—Section of free-discharge launder and Rheolaveur box.

Certain boxes on each launder are equipped for use with a vertical or counter current (Fig. 3), which allows only the materials of higher specific gravity to pass through. These boxes are used for the concentration and final drawoff of refuse.

Where the launder width exceeds 14 in., it is split by a partition and double boxes are used, one on each side of the partition (launder A, Table 1). Details of boxes are given in Tables 1 and 2.

Push water is introduced at the feed end of each launder, to convey the coal down the launder and to provide a medium for settling and classification over the boxes.

Flow of Coal and General Operation (Fig. 1)

All feed to the washing plant is minus $\frac{3}{8}$-in. material from the coarse-coal plant, which is raised to the top of the fine-coal plant by dewatering elevators. There it discharges into chutes delivering to the feed end of launder A. Launder B derives its feed from the material drawn off through launder A boxes. The same system is applied to launders C, D and E, where there is a continuous replacement by material drawn from the boxes directly above. Beds in all launders except A are disturbed by the discharge from the launders above them. Better washing results might be obtained by bringing the discharge of all the boxes from each launder back to the feed end of the launder below them but this would necessarily increase the head room and would necessitate longer elevators and a larger capital investment.

From the feed end to the overflow of each launder the ash content of the box discharges gradually decreases. The discharge of the first two or three boxes (the number depending on the character of feed)

from launder A is sufficiently high in refuse content to be delivered directly to the head of launder C instead of being fed into launder B.

The bed at the feed end of launder C is sufficiently heavy with impurities so that a small quantity of material sufficiently free of coal to be sent directly to refuse may be drawn off through the first box, provided a vertical current is applied to it. Whether or not this is done depends upon the quantity of high-ash, high-gravity material in the feed, and is left to the operator's discretion.

The first two boxes on launder D normally draw off refuse and use but little vertical current. The first box on launder E discharges clean refuse with the aid of a little vertical current and the last two boxes on launder E are used for drawing off fine refuse, always minus ⅛-in. and normally minus 1⁄16-in. size. This is accomplished by placing a 4-in. barrage ahead of the box and a 6-in. barrage after it, and bridging the two barrages with a section of ¼-in. perforated screen. The screen blinds sufficiently to pass only the very fine particles that were not drawn off through the other refuse boxes because of the vertical current rising in them. No vertical current is used on the fine boxes. Normally only one fine box is used but if the feed contains enough fine slate and pyrite to warrant it the second box can be used. The material drawn off through the fine boxes consists of fine slate, considerable calcite, some gypsum and a large amount of pyrite.

Close control of washing is maintained through visual inspection of float-and-sink tests made by the operator on small samples taken at various points. The tests are run in tall glasses of zinc chloride solution; clean coal in a solution of 1.40 sp. gr. and refuse in a solution of 1.60 sp. gr. If too much float coal is visible in the refuse samples the vertical current is increased or the size of the box-discharge opening is decreased. The quantity of allowable float in the refuse depends largely upon the ash desired in the clean coal.

The overflows from launders A and B go into clean metallurgical coal. The entire overflow of launder C may be diverted to clean coal, or entirely to recirculation, or the top of the launder overflow may be diverted to clean coal and the bottom portion of the overflow recirculated, the operation being termed "skimming." The overflows of launder D and launder E are recirculated.

Fine-coal Plant Circuits (Fig. 4)

The plant is designed so that one or more circuits can be used. The minus ⅜-in. from the coarse-coal plant can be sluiced to two feed-elevator boots, which we will designate as "east" and "west," and the distribution to each boot depends upon which flowsheet or method of washing is being employed at the time. Launders and gates throughout the plant are so arranged that any combination of material is available for each

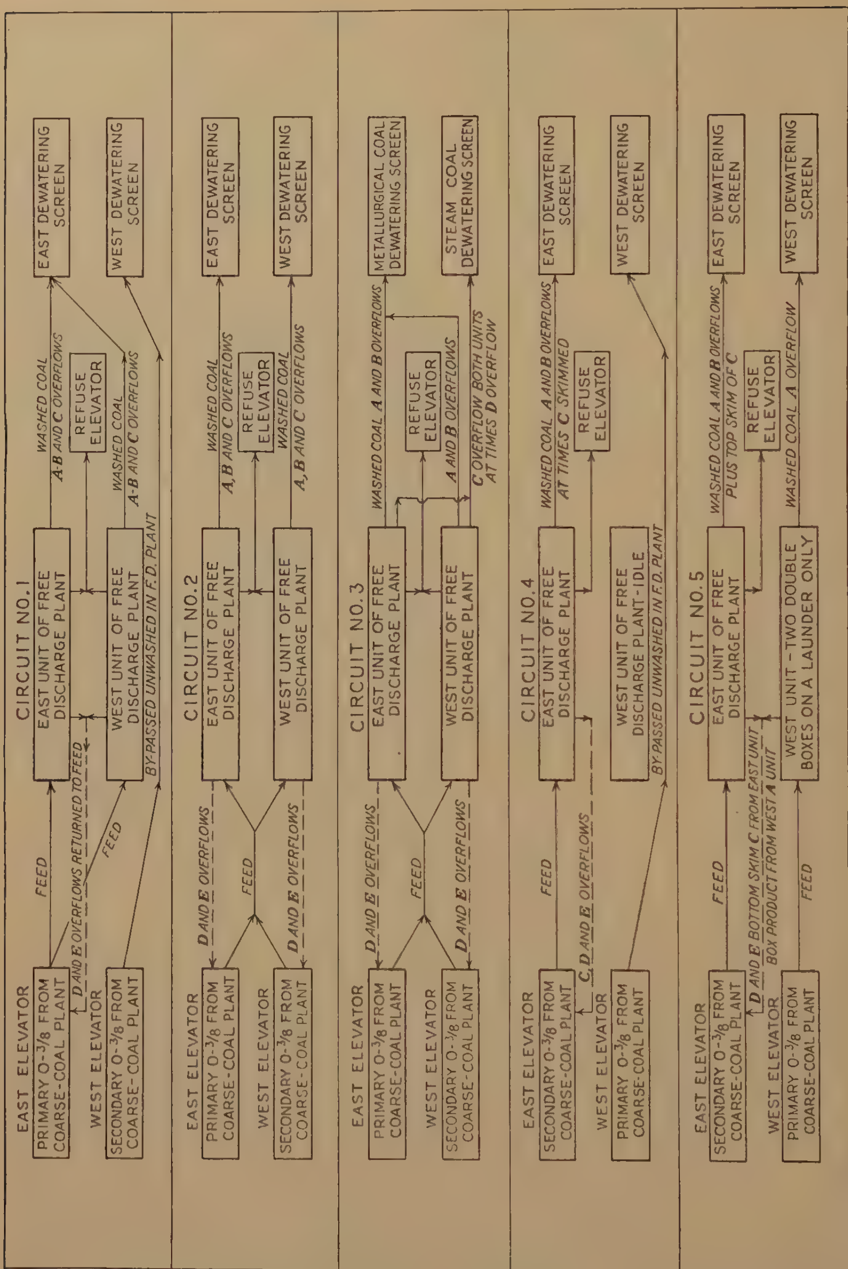

FIG. 4.—CIRCUITS OF FINE-COAL PLANT.

boot. The east elevator, however, always delivers to the feed of launder A of either unit. The west elevator may deliver to launder A of either unit or to a chute that sends the material to two wedgewire shaker screens, thereby by-passing the fine-coal plant.

Several circuits or methods have been used or tried for washing of the minus 3/8-in. coal in the fine-coal plant. Some of these have been used not alone for metallurgical reasons but somewhat because of market conditions either as regards ash and sulphur content or because of improvements thereby in the drying of the product after washing.

In a circuit that will be referred to as circuit No. 1 the minus 3/8-in. coal from the primary circuit* only of the coarse-coal plant was washed in parallel in both units of the fine-coal plant, making a three-product separation—metallurgical coal, middlings and refuse. The middlings were returned to the feed. A set of results obtained from this circuit are given in Table 3. In comparing these results with others, it should be remembered that by not including the minus 3/8-in. coal from the secondary circuit of the coarse-coal plant, the feed contained considerably less high-ash bony material than that which ordinarily is washed in the fine-coal plant. A, B and C launder overflows go to metallurgical coal; D and E launder overflows are recirculated.

Results as listed in Table 3 show that better results might be obtained by recirculating the product of launder C for the purpose of increasing the load and thereby increasing the depth of bed; a similar effect could be obtained by increasing the height of launder C barrages or narrowing launders B and C. Less drawoff from launders B and C would increase the ash content of the box products and together with increased vertical current on the refuse boxes of launders D and E would permit the removal of a better refuse. The refuse box of launder C could not be used because obviously the quantity of refuse was not sufficient to warrant a drawoff on that launder. About 45 tons of coal per hour exclusive of recirculating material was being fed to each unit during the test run as tabulated in Table 3.

With circuit No. 2 (Fig. 4) all minus 3/8-in. coal from all parts of the coarse-coal plant was treated in the fine-coal plant. Both units were run in parallel and refuse drawn from both. Three products (metallurgical coal, middlings, and refuse) were made. A, B and C launder overflows were sent out as metallurgical coal and the middlings, overflows from launders D and E to recirculation. Tables 4, 5 and 6 give a complete set of data on each box, and represent the average of both units run in parallel. During the test runs as tabulated in these tables about 55 tons per hour, exclusive of recirculating material, was being fed to each unit.

* J. T. Crawford, C. P. Proctor and M. J. Williams: Launder Washing of Coarse Coal. *Trans.* A.I.M.E. (1938) **130**, 172.

TABLE 3.—*Circuit No. 1, Screen Analyses and Ash of Products from Fine-coal Plant*

TYLER STANDARD SIEVES

Material	Head Ash	Plus 4-mesh		4 to 14-mesh		14 to 48-mesh		48 to 100-mesh		100 to 200-mesh		Minus 200-mesh	
		Size	Ash	Size	Ash	Size	Ash	Size	Ash	Size	Ash	Size	Ash
Feed, east unit...	7.6	15.4	6.0	51.3	5.9	26.7	9.3	4.6	13.2	0.5	15.4	1.5	15.8
Feed, west unit...	7.9	15.1	6.9	53.5	6.2	24.4	10.5	4.7	16.1	0.6	17.1	1.7	17.5
Overflow A, east unit...........	4.2	29.4	3.7	35.2	3.7	26.1	4.0	7.6	4.8	0.7	6.3	1.0	8.2
Overflow A, west unit...........	4.0	29.9	4.1	30.7	3.7	29.6	4.3	8.9	6.0	0.2	9.2	0.7	9.7
Overflow B, east unit...........	5.5	18.4	4.3	51.6	4.1	25.2	4.8	3.2	7.2	0.8	7.6	0.8	11.7
Overflow B, west unit...........	4.2	30.8	4.5	47.8	4.1	17.8	4.5	2.9	6.3			0.7	7.7
Overflow C, east unit...........	5.5	14.5	4.8	47.2	4.1	32.5	4.9	4.7	8.1	0.5	10.9	0.6	16.3
Overflow C, west unit...........	5.6	12.1	5.0	50.5	4.7	32.3	6.0	3.7	9.7	0.5	11.6	0.9	15.4
Overflow D, east unit...........	9.0	7.5	12.1	57.2	6.6	31.2	10.1	2.6	21.8	0.7	22.9	0.8	31.5
Overflow D, west unit...........	6.9	7.8	5.8	45.8	4.6	39.7	7.7	5.5	8.1	0.6	9.7	0.8	16.2
Overflow E, east unit...........	9.7	8.2	9.8	55.3	6.4	31.7	10.3	3.7	24.9	0.4	23.3	0.7	24.7
Overflow E, west unit...........	10.8	12.7	8.0	46.2	6.0	35.1	10.5	5.2	29.0	0.2	26.6	0.6	30.6
Refuse, east unit	30.6	18.1	32.5	47.1	29.7	31.8	36.6	2.6	53.7			0.4	41.5
Refuse, west unit	28.6	20.7	52.2	47.2	28.4	28.1	25.2	2.0	52.2	1.2	46.9	0.8	43.5

Circuit No. 3 is the circuit for which the plant was originally designed. All minus 3/8-in. coal was treated in the fine-coal plant both units running in parallel, each taking half the feed and making a four-product separation of metallurgical coal, steam coal, middlings, and refuse, middlings being returned to the feed. When using this circuit the overflow of launders A and B was metallurgical coal, that of C and occasionally of D was steam coal, and E was recirculated. When the product from launder D was not put out as steam coal, it likewise was recirculated. Tables 7 and 8 show a tabulation of results of test runs with circuit No. 3 at the time about 50 tons per hour, exclusive of recirculating material, was being fed to each unit.

Table 9 shows a complete analysis on minus 3/8-in. raw coal, washed coal and refuse, which perhaps better than any other set of data shows the results obtained when making three final products. (Circuit 3, Fig. 4.)

TABLE 4.—*Circuit No. 2, Average Screen Analyses and Ash of Products of Both Units of Free-discharge Plant*

TYLER STANDARD SIZES

Product	Percentages													
	Head Ash	⅜ in. to 4-mesh		4 to 14-mesh		14 to 48-mesh		48 to 100-mesh		100 to 200 mesh		Minus 200-mesh		Barrage, In.
		Size	Ash	Size	Ash	Size	Ash	Size	Ash	Size	Ash	Size	Ash	
Feed.......	9.0	16.8	7.9	46.6	8.1	31.4	10.6	3.3	13.5	1.5	14.6	0.4	17.5	
Launder A:														
Box 1..	15.0	11.1	14.5	47.5	13.9	36.8	15.6	3.1	18.1	2.1	20.6	0.4	20.3	2
Box 2..	13.8	16.7	13.6	52.0	12.8	27.5	15.2	2.0	15.8	1.5	21.3	0.3	19.0	2
Box 3..	15.3	10.1	14.1	50.8	13.2	34.7	18.0	2.7	23.4	1.4	26.0	0.3	21.9	2
Box 4..	10.7	9.6	11.1	51.6	8.1	34.7	13.1	2.5	18.8	1.2	20.0	0.4	18.2	2
Box 5..	8.5	11.5	8.5	51.8	6.9	32.5	10.3	2.3	15.4	1.5	22.6	0.4	17.4	2
Box 6..	7.0	13.0	6.8	51.4	6.0	31.4	8.0	1.9	14.9	2.0	14.9	0.3	18.4	2
Box 7..	6.6	12.6	6.5	50.2	5.3	32.9	7.7	2.5	13.5	1.5	20.6	0.3	19.9	2
Box 8..	6.3	16.2	5.7	51.5	5.2	28.8	7.0	1.8	10.0	1.4	19.1	0.3	17.0	¾
Box 9..	5.7	20.9	5.3	50.8	5.1	25.1	6.3	1.3	9.8	1.6	18.9	0.3	18.0	¾
Box 10.	No drawoff necessary													
Box 11.	No drawoff necessary													
Overflow[a]	4.2	29.0	4.2	37.0	3.7	29.7	4.4	2.4	6.2	1.6	7.0	0.3	13.9	
Launder B:														
Box 1..	21.4	10.7	22.8	52.4	17.2	33.4	23.4	2.0	31.9	1.2	28.0	0.3	19.1	2
Box 2..	16.9	6.5	21.2	49.5	12.9	39.8	19.8	3.1	25.7	0.8	37.8	0.3	19.6	2
Box 3..	11.3	7.8	13.7	47.7	7.8	39.6	13.4	3.2	22.4	1.4	31.4	0.3	21.8	2
Box 4..	8.8	6.3	11.1	50.4	7.2	39.4	9.0	2.7	15.9	0.9	18.6	0.3	17.0	2
Box 5..	6.0	15.0	6.8	58.6	5.0	24.2	6.3	1.1	11.0	0.8	14.5	0.3	16.7	2
Box 6..	5.9	13.5	6.0	56.9	4.8	27.3	6.7	1.4	12.8	0.8	17.1	0.2	16.3	2
Box 7..	5.2	19.2	5.3	53.2	4.6	24.9	5.5	1.3	10.7	1.0	11.9	0.4	13.9	2
Box 8..	Closed. No drawoff necessary													
Box 9..	Closed. No drawoff necessary													
Overflow[a]	4.5	27.2	4.7	43.5	4.3	26.2	4.5	1.0	6.5	1.8	7.3	0.3	13.0	
Launder C:														
Box 1[b].	Closed. No refuse drawn off													
Box 2..	35.0	10.2	49.3	48.7	34.5	37.0	31.6	2.4	33.8	1.4	38.7	0.3	22.6	2
Box 3..	26.5	11.0	42.5	56.6	20.3	28.6	25.7	0.7	29.6	2.7	33.7	0.4	21.9	2
Box 4..	21.9	10.6	27.7	50.4	16.3	34.8	21.5	2.2	28.8	1.7	30.0	0.3	21.0	2
Box 5..	15.9	11.0	18.2	50.7	11.3	34.4	18.0	2.5	24.1	1.1	33.1	0.3	21.9	2
Box 6..	14.5	6.3	18.6	46.8	11.5	41.6	16.7	3.7	24.3	1.2	37.0	0.4	21.7	2
Box 7..	13.0	3.6	21.8	43.6	12.8	46.2	17.5	4.0	27.4	2.2	31.9	0.4	24.5	2
Box 8..	Closed													
Overflow[a]	5.5	20.5	5.6	48.5	5.0	27.4	5.6	1.7	8.9	1.6	10.0	0.3	15.1	
Launder D:														
Box 1..	Closed. No refuse drawn off													
Box 2..	Closed. No refuse drawn off													
Box 3..	44.9	10.7	57.7	54.1	41.6	32.4	47.2	1.1	53.0	1.4	40.4	0.3	24 5	2
Box 4..	Closed													
Overflow.	14.8	7.3	14.2	44.6	9.9	43.4	19.9	2.5	31.0	1.9	26.6	0.3	17.5	
Launder E:														
Box 1[c].	66.5	22.7	66.1	65.1	65.2	11.8	69.8	0.1	42.0	0.1	42.6	0.2	31.7	2
Box 2[d].	58.2	0.0		41.0	58.2	53.4	58.8	4.4	53.0	0.6	35.1	0.6	27.9	6
Box 3..	Closed													
Overflow.	25.9	5.8	39.4	56.0	21.2	36.2	29.6	0.8	37.8	1.0	34.9	0.2	18.8	
Final refuse	66.3	20.9	66.0	60.2	66.9	15.4	69.0	0.9	51.0	0.7	57.1	1.9	53.6	

[a] Sampled ahead of stationary wedgewire screen, which further cleans the coal passing over it by removal of some minus 48-mesh particles with the water.

[b] No refuse drawn off as it was advisable to use it for holding the bed in E launder.

[c] Vertical current applied as needed to this box. Box product to final refuse.

[d] Fine refuse box. No vertical current used. Box product to final refuse.

TABLE 5.—*Circuit No. 2, Gravity Separation Characteristics of Feed and Final Products*

Material	Separation of ⅜-in. to 48-mesh[a]						0 to ⅜-in. Head Ash
	1.40 Float		1.40–1.60		1.60 Sink		
	Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash	
Raw coal	93.7	5.1	1.5	24.8	4.8	64.5	8.4
Metallurgical coal[a]	95.8	4.8	2.9	19.4	1.3	43.5	5.7
Free-discharge plant refuse	4.5	6.4	3.1	28.1	92.4	71.5	66.4

[a] Metallurgical coal separation made on 0 to ⅜ in., Table 6.

TABLE 6.—*Circuit No. 2, Size and Gravity Separation Characteristics of Metallurgical Coal*

Size, Tyler Mesh	Per Cent Size	Ash	1.40 Float		1.40 × 1.60		1.60 Sink	
			Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash
Minus ⅜-in. head	100.0	5.7	95.8	4.8	2.9	19.4	1.3	43.5
Plus 4-in.	15.5	6.0	96.2	5.2	2.9	21.3	0.9	40.8
4 to 14	58.6	5.6	96.3	4.8	2.6	19.9	1.1	40.9
14 to 48	22.3	6.0	94.6	4.6	3.3	18.9	2.1	49.1
48 to 100	1.7	5.7	93.5	4.5	4.4	15.9	2.1	39.0
100 to 200	0.7	5.4	90.5	4.2	7.3	15.0	2.2	24.4
Minus 200	1.2	6.1	91.6	5.2	6.2	13.9	2.2	20.4

It must be understood that where the analyses of an item in one table do not agree with the analyses of the same item in another table changes in the circuit, or coal fed to the plant, or grade of coal desired from the plant, caused the variation. All data in the body of this paper have been gathered over a period of six years of changing feed and practice.

In circuit No. 4 the primary minus ⅜-in. was by-passed and the secondary minus ⅜-in. treated in the fine-coal plant. This is directly opposite to circuit No. 1, in which the secondary minus ⅜-in. was by-passed and the primary minus ⅜-in. treated in the fine-coal plant.

The overflow of launders A and B and sometimes a portion skimmed from the top of launder C constitute the clean coal. All or only the bottom portion of the overflow of launder C and all the overflow of launders D and "E" was recirculated as rewash material. This served to iron out fluctuations in the feed to the plant. Launder E overflow was passed over a wedgewire dewatering screen, and the "through

material" from this screen was delivered to the refuse-elevator sump. The overflow of launder C carries a higher percentage of water and a finer coal than either launders A or B, thus rendering it exceptionally hard to dewater.

TABLE 7.—*Circuit No. 3, Ash of Feed and Products of Fine-coal Plant*
FEED TO FINE-COAL PLANT, 11 PER CENT OF ASH

Products	Percentage of Ash Content				
	Launder A	Launder B	Launder C	Launder D	Launder E
Box No.					
1	20.5	21.5	72.3[a]	77.6[a]	77.4[a]
2	14.0	17.2	42.1	Closed	65.7[a]
3	12.8	14.7	Closed	53.7	
4	11.5	10.6	39.5	50.1	
5	11.7	9.5	31.7		
6	8.1	8.0	27.3		
7	8.5	6.7	31.5		
8	7.5	8.0			
9	6.2				
10	Closed				
11	Closed				
Overflow	4.9	5.5	7.5	21.7	54.7
To rewash recirculation			12.2	21.7	54.7

[a] Products of first box off C launder, first box off D launder and both E launder boxes constituted the refuse from the fine-coal plant.

TABLE 8.—*Circuit No. 3, Gravity Separation Characteristics of Metallurgical and Steam Coal*

Material	Percentages						
	Head Ash	1.40 Float		1.60 Float		1.60 Sink	
		Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash
0 to ⅜-in. metallurgical	4.9	97.4	4.4	1.9	17.1	0.7	35.2
0 to ⅜-in. steam coal	12.5	76.2	4.7	10.6	17.0	13.2	53.6

Tables 10 and 11 list available data on circuit No. 4. For the test run using circuit No. 4 approximately 70 tons per hour exclusive of recirculating material was being fed to one unit only of the fine-coal plant.

For Circuit No. 5 (Fig. 4), the results obtained in one unit of the fine-coal plant at 80 tons per hour exclusive of the recirculating load are listed in Table 12. Approximately 110 tons per hour was the total load

TABLE 9.—*Circuit No. 3, Screen Analyses, Ash, Sulphur, Analyses of Ash, and Fusion of Feed and Plant Products*

	Size	Percentages			Ash Analysis, Per Cent				Ash Fusion, Deg. F.
	Inches and Tyler Mesh	Size	Ash	Sulphur	SiO_2	Al_2O_3	Fe_2O_3	CaO	
−3/8-in. raw coal...	Head	100.0	7.2	1.10	43.4	30.7	10.4	8.4	2300
	3/8-in. to 14 mesh	72.5	6.9	1.10	47.8	30.8	10.0	5.6	2430
	14 to 48 mesh	17.3	8.7	1.20	38.0	25.0	10.0	13.7	2290
	48 to 100 mesh	3.6	10.3	1.50	34.7	24.8	13.2	14.8	2230
	100 to 200 mesh	1.9	10.3	1.70	33.2	22.5	15.9	14.7	2220
	−200 mesh	4.7	12.1	1.30	31.5	21.6	13.4	16.1	2130
−3/8-in. metallurgical coal........	Head	100.0	4.9	1.00	48.4	35.7	8.5	4.3	2620
	3/8-in. to 14-mesh	74.4	4.9	0.95	49.3	37.1	7.5	3.4	2630
	14 to 48-mesh	20.9	4.7	0.95	47.4	36.0	8.4	4.9	2600
	48 to 100-mesh	2.4	5.2	0.95	43.7	32.1	9.9	7.4	2380
	100 to 200-mesh	0.8	5.3	1.05	43.3	32.6	12.5	8.0	2310
	−200-mesh	1.5	7.0	1.20	36.0	26.6	13.1	11.3	2330
−3/8-in. steam coal	Head	100.0	8.9	1.10	39.2	27.8	10.4	10.3	2290
	3/8-in. to 14-mesh	32.9	11.1	1.20	46.4	25.3	9.3	6.4	2200
	14 to 48-mesh	46.0	7.7	0.95	40.8	27.2	9.0	9.9	2300
	48 to 100-mesh	13.7	8.5	1.15	35.6	24.8	11.2	13.3	2220
	100 to 200-mesh	4.1	8.9	1.30	33.8	24.1	13.9	13.1	2180
	−200-mesh	3.3	12.0	1.30	31.8	20.3	13.3	15.1	2130
−3/8-in. refuse.....	Head	100.0	71.7	6.60	53.1	25.3	13.9	3.7	2310
	3/8-in. to 14-mesh	90.3	72.3	6.70	54.7	25.2	13.8	2.8	2360
	14 to 48-mesh	7.7	67.3	5.80	34.3	18.9	13.7	18.9	2110
	48 to 100-mesh	1.2	63.4	8.40	27.1	13.4	20.4	22.2	2070
	100 to 200-mesh	0.2	54.7	13.70	21.9	15.2	37.1	15.4	2150
	−200-mesh	0.6	38.5	9.80	31.7	16.8	27.6	11.2	2140

when the feed consisted only of the minus 3/8-in. coal from the secondary circuit of the coarse-coal plant. For this test run the secondary minus 3/8-in. coal is shown under "Feed, East Unit." The minus 3/8-in. coal from the primary circuit of the coarse-coal plant is shown under "Feed, West Unit" and was washed over two double boxes only in launder A west unit. The drawoff of these two double boxes was returned to the east elevator with the feed to east unit. The minus 3/8-in. clean coal comprises the overflow of launder A west unit plus the overflow of launders A and B and the top skimming of launder C east unit.

Discussion of Launder Washing

Opinions as to the importance of the beds in the launders of the free-discharge plant are varied. In the writers' opinion, when the heads of the beds are held far back along the classification section, resulting in deeper beds for the entire length of the launder, though much less water

TABLE 10.—*Circuit No. 4, Screen Analyses and Ash of Fine-coal Plant Products*

Sample	Percentages												
	Head Ash	⅜-in. to 4-mesh		4 to 14-mesh		14 to 48-mesh		48 to 100-mesh		100 to 200-mesh		−200-mesh	
		Size	Ash	Size	Ash	Size	Ash	Size	Ash	Size	Ash	Size	Ash
West elevator discharge (to clean coal)	6.0	17.5	6.0	44.0	5.7	25.5	6.1	8.5	6.4	2.0	13.7	2.5	15.2
East elevator (feed to one unit fine discharge)	12.9												
Overflow of launder A	5.5	48.0	5.7	38.0	5.0	10.0	4.9	2.5	4.6	0.5	6.8	1.0	12.1
B	6.9	10.3	9.6	51.0	6.8	33.0	6.3	3.5	8.5	1.0	13.3	1.0	15.1
C	9.0	3.5	8.5	32.0	5.8	47.5	7.8	1.5	18.2	3.5	40.0	2.0	34.3
D	13.1	2.5	17.6	36.0	9.1	49.0	12.3	8.3	25.7	2.5	47.4	1.5	31.4
E (before dewatering)	25.6	7.0	39.3	45.0	22.4	43.0	24.0	4.0	53.1	1.0	45.5	(−100-mesh)	
E (after dewatering)	23.8	5.5	38.7	44.0	20.9	46.0	22.5	4.0	47.1	0.5	44.5	(−100-mesh)	
E (dewaterings through)	44.7			1.5	38.3	13.0	36.1	7.0	51.9	38.5	37.1	(−100-mesh)	

TABLE 11.—*Circuit No. 4, Gravity Separation Characteristics of Fine-coal Plant Launder Overflows*

Sample	⅜-in. to 28-mesh								
	Head Ash	1.40 Float		1.60 Float		1.60 Sink		−28 Mesh	
		Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Size, Per Cent	Ash, Per Cent
Overflow of launder A	5.5	98.5	5.2	1.2	22.3	0.3	30.4	10.1	7.0
B	6.0	97.5	5.5	2.2	22.9	0.3	36.8	17.7	8.2
C	7.9	89.5	5.6	8.3	23.2	2.2	42.0	15.3	17.4
Skimming of launder C	6.7	94.5	5.6	4.4	22.3	1.1	38.4	12.0	12.7
Overflow of launder D	13.3	76.0	5.5	10.5	24.6	13.5	48.3	20.6	34.7

is used and the launder presents a smooth and nice looking surface, the ash in the clean coal is invariably higher. This opinion is based upon the actual running of the plant, visual sink-and-float tests on all the boxes and upon appearance of the launder overflow and box-discharge products.

TABLE 12.—*Circuit No. 5, Size and Gravity Separation Characteristics of Fine-coal Plant Products*

Size, In., and Tyler, Mesh	Percentages							
	1.40 Float			1.40–1.60		1.60 Sink		Head Ash
	Size	Wt.	Ash	Wt.	Ash	Wt.	Ash	
Feed to East Unit								
⅜-in. to 4-mesh	20.5	82.7	6.1	12.1	25.9	5.2	51.6	10.9
4 to 8	32.3	80.8	6.1	12.2	25.6	7.0	55.8	12.0
8 to 14	24.3	79.7	5.8	12.3	24.1	8.0	57.6	12.2
14 to 48	22.9	82.2	5.3	6.9	22.8	10.9	60.5	12.5
⅜-in. to 48-mesh	100.0	81.3	5.8	10.5	24.4	8.2	56.8	12.0
Launder A Overflow, East Unit								
⅜-in. to 4-mesh	26.4	98.5	5.1	1.5	23.7	0.0		5.4
4 to 8	21.0	98.2	5.6	1.7	23.5	0.1	45.2	5.9
8 to 14	13.5	97.6	4.9	2.0	23.7	0.4	39.7	5.4
14 to 48	39.1	98.2	5.5	0.7	20.5	1.1	36.3	5.9
⅜-in. to 48-mesh	100.0	98.2	5.3	1.3	23.2	0.5	37.0	5.7
Launder B Overflow, East Unit								
⅜-in. to 4-mesh	22.3	92.4	6.7	7.2	22.4	0.4	38.1	7.9
4 to 8	35.0	94.6	6.0	4.6	21.8	0.8	38.8	7.0
8 to 14	18.6	94.6	5.4	4.0	21.9	1.4	41.5	6.6
14 to 48	24.1	93.3	4.5	4.8	16.8	1.9	41.4	5.9
⅜-in. to 48-mesh	100.0	93.8	5.7	5.1	20.8	1.1	39.6	6.9
Launder C Overflow, East Unit								
⅜-in. to 4-mesh	8.2	84.6	6.9	13.6	24.9	1.8	43.5	10.0
4 to 8	25.8	92.1	5.4	6.5	24.5	1.4	40.6	7.1
8 to 14	25.5	93.4	5.2	4.7	24.2	1.9	44.1	6.8
14 to 48	40.5	92.1	4.9	4.2	22.7	3.7	45.7	7.1
⅜-in. to 48-mesh	100.0	91.8	5.3	5.7	24.7	2.5	43.8	7.3
Launder D Overflow, East Unit								
⅜-in. to 4-mesh	8.7	72.3	7.6	17.2	26.6	10.5	40.7	14.4
4 to 8	24.3	82.1	6.4	9.4	25.5	8.5	42.8	11.3
8 to 14	25.7	85.7	5.8	6.4	23.6	7.9	47.3	10.3
14 to 48	41.3	84.8	5.8	4.1	22.5	11.1	53.2	11.7
⅜-in. to 48-mesh	100.0	83.3	6.1	7.1	23.6	9.6	48.0	11.4
Launder E Overflow, East Unit								
⅜-in. to 4-mesh	25.9	81.6	9.0	6.8	24.8	11.6	42.1	13.9
4 to 8	36.3	67.7	6.9	12.6	24.0	19.7	45.7	16.7
8 to 14	22.8	53.8	6.3	21.2	23.2	25.0	52.0	21.3
14 to 48	15.0	33.8	6.4	42.8	21.5	23.4	61.9	25.8
⅜-in. to 48-mesh	100.0	63.1	7.1	17.6	23.6	19.3	49.8	18.2
Feed to West Unit								
⅜-in. to 4-mesh	26.1	96.7	5.6	2.4	24.0	0.9	46.5	6.4
4 to 8	27.7	96.1	4.8	2.7	22.5	1.2	49.3	5.8
8 to 14	20.4	95.5	5.1	2.7	21.9	1.8	49.8	6.4
14 to 48	25.8	92.2	4.6	2.6	19.4	5.2	45.7	7.1
⅜-in. to 48-mesh	100.0	95.1	5.0	2.6	22.0	2.3	47.6	6.4
Launder A Overflow, West Unit								
⅜-in. to 4-mesh	30.6	97.3	5.3	2.2	23.0	0.5	42.0	5.0
4 to 8	28.3	96.7	5.2	2.5	22.4	0.8	45.2	6.0
8 to 14	18.7	95.0	5.1	3.8	22.3	1.2	46.8	6.3
14 to 48	22.4	93.8	4.5	3.7	19.4	2.5	47.5	6.1
⅜-in. to 48 mesh	100.0	95.9	5.1	2.9	21.9	1.2	44.9	6.1
Clean Coal from Both Units								
⅜-in. to 4-mesh	20.5	95.8	6.2	3.7	22.6	0.5	42.0	7.0
4 to 8	32.3	96.0	5.2	3.3	22.8	0.7	42.6	6.1
8 to 14	24.3	96.2	5.1	2.9	23.4	0.9	38.8	5.9
14 to 48	22.9	93.7	4.6	4.3	17.9	2.0	46.2	6.0
⅜-in. to 48-mesh	100.0	95.5	5.3	3.5	21.7	1.0	42.4	6.3
Refuse								
⅜-in. to 48-mesh	100.0	1.9	7.0	1.9	29.2	96.2	70.4	68.4

After the coarse refuse is removed the smaller refuse tends to settle out more quickly as the material passes along the launders. The coarse sizes of coal remain near the surface, with the result that not until the last few boxes of each launder are reached does much coarse coal pass through the boxes. The settling of fine refuse is somewhat hindered in the lower launders by the disturbance in the upper part of the bed caused by the discharges from the boxes above, hence if the discharge of all boxes from each launder were returned to the head of the lower launder instead of being run on top of the bed, as in present practice, it is probable that better washing should result. This, however, would necessitate considerable increase in height of the fine-coal plant.

When sufficient push water is used the beds are shallower and much less compact and each particle becomes less intimately associated with other particles, thus giving the high-gravity material from the beds above a much better chance to classify in time to pass through the boxes before it passes off in the launder overflow. In this case, the launder surfaces generally appear to be rough and require more attention from the operator to prevent the beds from becoming (especially launder B bed) lost entirely. Washing with more water also results in classification with regard to specific gravity, whereas with less water the coal tends to classify more with regard to size. In one case the grade of coal is better but there is more water to handle; in the other, the washing is less efficient but there is less water to handle.

The main effort in any system of launder washing eventually resolves itself into a continuous attempt to get the material in the launders to classify and bed according to specific gravity. Unfortunately coal tends to classify and bed according to size rather than according to specific gravity, particularly where the material flows sluggishly. Sufficient water must be used to keep the bed liquid and the particles separated as much as possible. Vertical current water through the boxes or other slots in the bottom of the launders for the purpose of loosening the bed above the boxes is often beneficial. Raising barrage heights and increasing the amount of push water at the heads of launders, with a resulting loosening of the bed, has in several instances been successful in reducing the amount of 1.60 sink material in the clean coal from 1.0 to 0.4 per cent.

Depth of bed in a launder is controlled by the amount of drawoff from the launder above, the amount of drawoff from the launder itself, the amount of push water used, and the height of the barrages. Amount of vertical current used on any box is likewise a contributing factor.

Push water on launders B, C, D and E is seldom varied; the beds in these launders are controlled mainly by the push-water valves of launder A and by regulation of the box discharges. Variation in the bed of

launder A caused by variations in the feed or amount of push water used is quickly transmitted to the beds in the launders, because a variation in launder A bed varies the amount passing through the launder A boxes.

In general, launder length should be increased with any increase of sink material in the raw feed. If the finer portions of the feed contain the bulk of the refuse to be removed it may be advantageous to bed and classify as to size; in this case deep sluggish beds may be beneficial.

The so-called "classification" sections at the head ends of launders serve primarily as chutes for starting the material on its way, as no classification occurs until the material slows down in the sections that have flatter pitch. For this reason a flat-pitch section is desirable between the end of the "classification" pitch and the first Rheo box.

Launder C refuse box is the most difficult to control. If carefully watched it is possible to draw off a product containing a high percentage of free impurities. Because of this, the beds in launder C require more attention than those of launders A and B. Launders D and E require more water proportionally than launders A, B and C because of the higher specific gravity of the material, even though the tonnage is less. The first box off launder D is usually set to draw a clean refuse with little or no float at 1.60 gravity. The second box of launder D passes more fines than the first box but unless sufficient vertical current is used too much float coal is removed and, in addition (which is most important), insufficient high ash fines are left for the fine box of launder E. Most of the refuse control comes from the first box of that launder. The bed from which this box draws derives its material from the third and fourth boxes of launder D. Increases of the vertical current in the first box of launder E increases the ash content of the launder E overflow. This returns more free slate to rewash, and thence to the feed to the plant, resulting in an improved refuse product from the two launder D boxes should they be pulling too much 1.60 float material.

If the first two boxes of launder D and the first box of launder E are set correctly, a sufficient quantity of high-gravity fines is present in launder E at the second or fines removal box, so that a refuse of better than 65 per cent ash can be discharged by this box (E-2, Fig. 1) without the use of any vertical current (a highly desirable feature). Variation of the vertical current on the second launder D box usually results in poor operation of the launder E fine box. After becoming familiar with the plant an operator can do most of the control work from the push currents on launder A and the vertical current on the first box of launder E. Launder E push water is seldom changed and becomes a control indicator in that a thickening of the launder E bed means that insufficient refuse is being drawn off, and vice versa. As the smaller sizes (minus 28) are the "dirtiest" it may easily be seen why operation of the fine box is important.

Reduction of the ash content in the refuse below a certain amount does not cause a corresponding reduction in the ash content of the cleaned coal. If the classification and bedding in the launders were perfect, or more nearly so than actually they are, an increased drawoff from the refuse boxes with a corresponding reduction in refuse ash content would result in a lowering of the 1.60 or 1.40 sink content of the cleaned coal. The actual bedding and classification are not perfect and it has been found that turning the entire overflow of launders D and E (for test purposes) out to refuse along with the drawoff of the refuse boxes has not caused any appreciable reduction of the sink content of the cleaned coal.

At the present time the total tonnage washed in one unit of five launders varies between 50 and 90 tons per hour of minus ⅜-in. coal. The tonnage overflowing launders D and E varies little. The overflow of launder E varies between 5 and 8 tons of solids per hour while the overflow of launder D varies from 15 to 20 tons of solids per hour.

Sulphur is removed as a normal consequence of removal of the high-gravity particles. Sulphur contents of raw coal, clean coal and refuse amount to about 1.10 per cent, 0.90 per cent and 3.5 per cent, respectively, at the present time.

Increase in the percentage of solids of the circulating water used for push water and vertical currents in the fine-coal plant over 5 per cent solids is usually detrimental to the washing, as thick water offers increased resistance to the settling out of the high-gravity particles and noticeably increases the sink content of the cleaned coal, particularly in the finer sizes under 14-mesh.

Table Washing as an Auxiliary to the Free-discharge Units

Continued increase (due to change in mining methods) in the amount of minus ⅜-in. coal to be cleaned finally made it necessary to increase the capacity of the fine-coal plant either by additional launders or auxiliary units. Final determination to use tables was based on the desire to reduce (as in the coarse-coal plant) the recirculation of the finer sizes, the build-up of which in the circuit was detrimental to the removal of the large particles of high-gravity material. Consideration of all factors resulted in the installation of three 7 by 14-ft. Plato concentrating tables.

For the table circuit the combined overflow of launders D and E is screened over a single-deck vibrating screen at 6 mesh, the oversize of which is recirculated to the head end of launder A, the undersize going to a small settling cone from which the minus 6-mesh material is raised and dewatered by a chain-and-bucket elevator. The elevator discharges to a feed box that distributes evenly to the three tables. The tables make a three-product separation: concentrates joining the overflow of launders A, B and C; middlings returning with the feed to the tables; tailings being laundered to the refuse elevators.

TABLE 13.—*Gravity Separation Characteristics of Plato Table Feed and Products*

Sample	Splits, Ft.	Plus 48-mesh				Calculated Head Ash, Per Cent	Percentage of Input
		1.60 Float		1.60 Sink			
		Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent		
Feed....................		62.7	9.8	37.3	62.2	29.4	100.0
Clean coal concentrate:							
Zone A................	11-side	98.4	8.6	1.6	41.0	9.1	49.4
Middlings:							
Zones B, C, D.........	2½-side 1-end	60.4	14.3	39.6	43.7	26.0	19.7
Zone B................	1-side	78.2	13.2	21.8	40.7	19.2	5.5
Zone C................	1-side	72.8	14.1	27.2	40.0	21.2	3.8
Zone D................	½-side 1-end	46.4	15.3	53.6	45.2	31.3	10.4
Refuse tailings:							
Zones E, F, G, H......	6-end	8.7	20.6	91.3	72.7	68.2	30.9
Zone E................	½-end	33.9	17.5	66.1	46.5	36.7	2.6
Zone F................	½-end	27.7	20.5	72.3	50.0	41.8	3.2
Zone G................	1-end	10.8	22.9	89.2	55.1	52.1	6.1
Zone H................	4-end	1.4	25.3	98.6	81.8	81.1	19.0
Calculated over-all head..		63.1	10.1	36.9	65.8	30.7	100.0

Feed, Separation at 1.40 and 1.60 Sp. Gr.

1.40 Sp. Gr.				1.40–1.60		1.60 Sp. Gr.				Head Ash, Per Cent	
Float		Sink				Float		Sink			
Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Calculated	Head
50.5	6.4	49.5	50.9	12.2	24.0	62.7	9.8	37.3	62.2	29.4	33.4

During these tests the following data on these tables remained the same:

Feed............... 5.0 tons per hour per table at 40.7 per cent solids.
Speed............. 275 r.p.m.
Stroke............ $1\frac{5}{16}$ in.
Transverse slope... ¼ in. per foot at feed end.
Longitudinal slope:
  Concentrate side, slight positive slope (⅜ in. in 13 ft.)
  Wash-water side, refuse corner raised ⅞ in.
Deck surface, corrugated rubber.

As in the coarse-coal plant, the elimination of part of the recirculation of fine coal measurably increased the washing action in the launders of the fine-coal plant and at the same time permitted an increase in the total tonnage of feed to the plant.

Many small changes were necessary in the surface of the tables in order to achieve the desired results. The original riffle spacing of 1½-in. caused "packing," so that an increase to 2-in. spacing was found necessary. With the original deck surface, the following results were obtained at 5 tons of feed per hour per table:

	1.60 Sink, Per Cent	Percentage of Input
Feed	23.5	100.0
Concentrate	2.1	54.5
Middlings	30.4	27.3
Tailings	92.1	18.2

By moving the Plato section (rise in deck) 12 in. toward the refuse end from its original position, a decided improvement was noted as below:

	1.60 Sink, Per Cent	Percentage of Input
Feed	37.3	100.0
Concentrate	1.6	49.4
Middlings	39.6	19.7
Tailings	91.3	30.9

The details of this separation are shown in Table 13.

Acknowledgment

To Mr. J. B. Morrow, Production Vice President of the Pittsburgh Coal Co., for his many helpful suggestions and permission to use the data incorporated herein, and to numerous others of the Production Department, Pittsburgh Coal Co., for their aid in obtaining the test data, the authors make grateful acknowledgment.

DISCUSSION

(H. F. Hebley presiding)

J. Griffen,* Pittsburgh, Pa.—When it is realized that the paper by Crawford, Proctor and Younkins is a summary of six years operation, during which some two million tons of minus ⅜-in. coal was handled, while changing mining methods materially modified the quality and characteristics of the raw material and the market imposed even more stringent demands for an improved product, it will be appreciated

* Koppers-Rheolaveur Co.

how difficult a problem faced the authors. I saw the first rough draft some time ago and the wealth of test data was somewhat bewildering.

Too much credit cannot be given to the management and operating personnel of the Pittsburgh Coal Co. for the resourcefulness and energy that determined the facts and devised the changes in the plant to meet the changing conditions and demands. The variety of plant circuits described indicates the flexibility of the launder system of washing and the ability to utilize this flexibility to meet changing demands on the washing plant. Other features of the launder system that might have been mentioned as having a definite influence on the costs of cleaning fine coal, are:

1. High capacity per unit of building space occupied.
2. Less water circulated per ton of coal handled, which considerably reduces the cost of water clarification, circulation and slurry recovery.
3. Washing units of high capacity, which reduce the cost and difficulty of delivering the raw feed and collecting the finished products.

The wet tables mentioned in the paper as used to supplement the launders are probably the only equipment in general commercial use that could replace the launder system and duplicate its cleaning performance. Probably 14 or 15 wet tables would be required to handle the 110 tons per hour treated by the two launder units. These tables would require floor space of about 3600 sq. ft. while the launder units occupy a space 14 ft. by 90 ft., which is about two floors high, or a total of 2520 sq. ft. The wet tables would require one-fourth to one-third more water than the launders.

Many of the features of the launder plant described are peculiar to the particular type of coal being cleaned. Ordinarily a fine-coal launder unit consists of only four superimposed launders instead of the five here described. Generally, the greater the amount of refuse in the raw coal, the less necessity for an extended progressive concentration of refuse.

In a launder system, as in other cleaning equipment, separation as to size is going on coincidentally with separation according to specific gravity. The coal product from the top (A) launder is coarser in size than that from the next lower (B) launder. Rheo boxes using vertical current discharge coarser refuse than those without vertical current. With such a wide range of sizes handled and cleaned, ⅜-in. rd. to 200 mesh or roughly 120 to 1, it is natural that difficulties should arise because heavy-gravity fine refuse is imperfectly separated from coarse lighter bone. In this plant improved results, and particularly greater capacities, were obtained by collecting such a mixed product from the launder system, screening it at 6 mesh, returning the plus 6 mesh to the launders and treating the minus 6 mesh on wet tables. In Europe, where the coals generally have much more near-gravity coarse refuse than the coal from the Pittsburgh seam, this problem is satisfactorily solved in launders by retreating the coarse and fine portions of such a mixed product in separate launder units.

The author's discussion of launder washing is very informative and the statements made are in general confirmed by my experience with launders on a variety of coals. Certain comments, however, may be of value.

On page 272 and again on page 283 it is suggested that "the beds in all launders, except (A), are disturbed by the discharge from the launders above them and better washing results might be obtained by bringing the discharge of all the boxes from each launder back to the feed end of the launder below them." This arrangement has been tried, but its supposed advantages overlook the fact that it concentrates the entire tonnage of refuse to be treated in a launder at the feed end of that launder, which interferes with perfect stratification and causes a deep refuse bed over the first boxes. In other words, it would cause a condition similar to that described at the bottom of page 280. With a raw coal containing only a small amount of refuse,

the suggestion might prove slightly advantageous. With coal having a refuse content of 5 per cent or more, the conventional arrangement has definite advantages as to capacity and separation which far outweigh the slight disturbance due to the discharge into a lower launder of material from the boxes on the launder above.

The various comments on page 283 relative to the control of launder classification may be summarized to the requirement that enough water must flow in the launder to produce full mobility of the individual particles and this condition is not obtained when the top of the stream looks smooth and regular, but on the contrary, is indicated by an appearance of roughness or slight turbulence.

In regard to the statement on page 284 that "in general, launder length should be increased with any increase of sink material in the raw feed," our experience, working with a great variety of coals, many of which contained much more refuse than the Pittsburgh seam, clearly shows that launders should not necessarily be lengthened to take care of increased amounts of refuse. Rather, that a further controlling factor is the tonnage of refuse handled per unit of width. The launders must be designed and arranged so that a certain maximum value is not exceeded at any point in the unit.

Flocculation and Clarification of Slimes with Organic Flocculants

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(New York Meeting, February 1939)

The application of wet cleaning processes for the beneficiation of bituminous coal has created in some localities a problem in the recovery and disposal of fine solids in the washery water. The maximum allowable concentration of these solids in the washery circuit that is consistent with satisfactory operation of the cleaning units remains a controversial point. It has, however, been generally recognized for a number of years that some degree of control should be exerted over the concentration of such material.

Early methods of control involved intermittent or continuous discharge of a portion of the plant slurry into near-by streams or settling ponds, and this persisted even after the adoption of settling tanks or cones. The practice is not always economical, for two reasons: first, because of the waste of coal, and, second, because it is sometimes difficult and expensive to obtain an adequate supply of water. The latter condition often is aggravated by chemical treatment of the make-up water that is necessary to prevent excessive corrosion of plant equipment. The modern approach to this problem has been the adoption of continuous thickening devices. However, because such equipment occupies considerable plant space and is somewhat expensive to construct, the practice of loading these thickeners beyond their rated capacity is widespread, therefore often the performance of such units is not altogether satisfactory.

In recent years, European washeries have increased the rate of settling of the fine coal from the slurry by the addition of specially prepared starch solutions. These reagents cause flocculation of the solid particles and thereby increase the rate of sedimentation.

The first important work on the flocculation and clarification of coal slurry was done by Henry, who developed the Henry process and used it successfully in Belgian preparation plants. Soon after the initiation in Belgium, the process was introduced into England, where subsequent investigation developed similar processes, which were successfully used in

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English plants. English investigators, including Raybould, Samuel, Wilkin, Needham, Holmes, and others, studied various phases of the problem.

In 1929, P. S. Gardner, Sr., of Koppers-Rheolaveur Co., who had seen the work in Belgium, suggested a plant test at Champion No. 1 Preparation Plant of the Pittsburgh Coal Co., using frozen starch and caustic. Immediately following this, P. S. Gardner, Jr., conducted tests on settling solids from washery water in a 12-ft. diameter settling cone at the Champion No. 1 plant. The results of the tests in which frozen starch was used were not wholly satisfactory, owing to difficulty in maintaining uniform flocculation. C. Q. Campbell, of Koppers-Rheolaveur Co., became interested in the possibilities of flocculation of coal slurry by means of starch treatment, and late in the year 1930 a full-scale plant test was made at Champion No. 1 plant using a reagent prepared by causticizing heated starch. The results of this test were reasonably satisfactory and showed clearly the possibilities of starch treatment for clarifying washery water.

In 1933 because of increased hourly tonnage, a higher percentage of fines in the raw coal, and a change in the characteristics of these fines, the problem of clarifying washery water became more serious than before. The Pittsburgh Coal Co. then started more laboratory and plant tests. S. B. Barley and William Robert tested a wide range of reagents, and starch was found to be one of the most effective in plant and laboratory tests.

While British and other European investigators have described and discussed the advantages of organic flocculants from an operating viewpoint, little attention has been given to the variables involved in the preparation of the starch reagent. This laboratory investigation into the preparation and properties of organic flocculating reagents was carried on by C. G. Black from February to July 1936, and by the authors from July 1936 to the middle of 1937.

There has been a twofold purpose in the present study: (1) to develop methods by which cheap materials could be used as flocculating reagents, and (2) to secure information on the flocculation process. A wide range of materials has been investigated and many of the variables that affect flocculation have been studied.

It has been found that the capacity of existing settling equipment can be markedly increased by the use of flocculating reagents, and their use is now a regular part of the plant operation.

Experimental Methods

A review of the literature disclosed a number of methods for obtaining data relative to flocculation and sedimentation of the solids present in a coal slurry. Most previous work has been conducted in glass cylinders

or similar types of vessels. With this type of container it is often difficult to observe the line of demarcation of the solids as settling occurs. Some investigators have found it necessary to measure changes in the concentration of solids due to sedimentation by withdrawing small volumes of the slurry as settling proceeded. Such a practice is obviously cumbersome and lengthy. To obviate these difficulties, a modification of the Henry settling frame was adopted for this investigation.

The settling frame consists of two glass plates, 7 by 19 in., held about ¼ in. apart by means of a rubber hose threaded with soft soldering wire. The plates are held in position by two wooden frames, connected by stove bolts ¼ in. in diameter. The distance between the glass plates is adjusted so that a volume of 500 ml. of slurry occupies a height of about 40 cm. The frame is lighted from the rear, with a reflector covered with a frosted glass plate, and a centimeter scale is fastened to the front glass of the frame.

The following procedure was followed in all tests: A 25-gram sample of dry solids was mixed with 100 ml. of water and allowed to stand overnight to insure thorough wetting of all particles. The sample was diluted to 500 ml. and the temperature adjusted to 29° C. The flocculating reagent was added and the sample poured back and forth between two beakers six times. It was then poured into the settling frame through a flat funnel. The time in seconds for each 4-cm. drop of the dividing line between solid and liquid was recorded until the zone of compression was reached. During the test the light was placed about one foot to the rear of the frame and the eyes of the observer were kept on a level with the line of demarcation of the solids.

Method of Recording Data

It has been found convenient to express the results of a test in the settling frame by means of the "settling index"; that is, the number of seconds required for the solids to settle a distance of one centimeter in the zone of free settling. For a given slurry it is a measure of the degree of flocculation of the particles. The lower the index, the greater the rate of sedimentation and the degree of coagulation. No difficulty will be experienced if the reciprocal relationship between rate of sedimentation and the settling index is kept in mind.

TABLE 1.—*Comparison of Settling Index with Clarification Produced*

SLURRY, 5 PER CENT SOLIDS

Settling Index, Sec. per Cm.	Clarification
21 or greater	Very poor
11 to 20 incl	Poor
6 to 10 incl	Fair
5	Good
4 or less	Very good (practically complete)

Table 1 shows the general relationship that has been observed between the "settling index" and degree of clarification with a slurry containing

5 per cent solids. This relationship does not necessarily hold true for slurries having a higher percentage of solids.

In the procedure described for determining the "settling index," it is possible to duplicate the results to within 10 per cent.

In order to facilitate examination of tabulated data, the treatment used in many tests has been reported on a dry basis both as parts per million and as pounds of reagent per ton of solids. To convert from pounds per ton of solids to parts per million, multiply by five times the percentage of solids.

Analyses of Solids and Water Used in This Investigation

Solids used in preparing standard laboratory slurries were obtained by filtration of the feed to a continuous thickener at the Champion No. 1 preparation plant. Tables 2 and 3 show the analyses of the solids and water used in preparing laboratory slurries. Table 2 shows that 64.6 per cent of these dried solids pass a 200-mesh screen, while 82 per cent pass a 100-mesh screen.

These solids constitute but one-third of the total solids in the washery slurry but are the most difficult portion to settle. A large volume of slurry is fed into a continuous thickener 60 ft. in diameter, and a classification of the solids occurs. The underflow contains the larger coal particles and to a certain extent the fine pyrite, and the overflow contains a high percentage of minus 200-mesh material, including most of the clay. Two-thirds of the total solids is settled in the 60-ft. thickener. Part of the 60-ft. thickener overflow and all of the refuse boot overflow form the feed to the 85-ft. diameter thickener and the solids from this product were used in the investigation. An untreated slurry of these solids has a "settling index" of 65.

Table 2.—*Analysis of Solids from Continuous Thickener Feed*

Size	Per Cent	Ash	Sulphur	Specific Gravity
Head	100.0	28.6	3.2	1.60
+28 mesh	0.8	29.9	3.4	1.73
28–48	3.2	16.0	2.6	1.49
48–60	2.5	10.3	2.0	1.37
60–100	11.5	18.8	2.4	1.46
100–200	17.4	25.6	3.1	1.46
−200 mesh	64.6	32.8	3.4	1.66

It is of interest to note in Table 3 the marked increase in the mineral content of the plant slurry filtrate over that of the make-up water. It should be emphasized that there was practically no waste of water from the plant at the time these samples were taken. All filtrate water was

being recirculated, and only sufficient make-up water was added to compensate for normal losses due to water carried out on the coal and refuse and to heat-drying losses.

TABLE 3.—*Comparison of Analysis of Water Used in Preparation of Laboratory Slurries with Plant Slurry Filtrate*

	Make-up Water	Plant Slurry Filtrate
pH (potentiometric)	7.2	8.3
Titrated alkalinity (methyl orange), p.p.m. Na_2CO_3	159.0	170.0
Mineral content, p.p.m.:		
SiO_2	0	0
Al_2O_3-Fe_2O_3	2	7
CaO	47	259
MgO	14	46
$SO_4^{=}$	58	710
Cl^{-}	7	493

EFFECT OF PERCENTAGE OF SOLIDS ON SETTLING INDEX AS DETERMINED IN SETTLING FRAME

As the percentage of solids in the slurry increases, the rate of settling decreases. This effect is independent of the material or the treatment used. It is possible by a few tests to determine for any given material and treatment the percentage of solids that will give the best settling.

TABLE 4.—*Effect of Percentage of Solids on Settling Index and on Point of Compression in Settling Frame*

SLURRY TEMPERATURE, 29° C.

Centimeters	Settling Index, Sec. per Cm.							
	Untreated Slurry				Treated Slurry[a]			
	5 Per Cent Solids		10 Per Cent Solids		5 Per Cent Solids		10 Per Cent Solids	
	4-cm. Drop	Cum.	4-cm. Drop	Cum.	4-cm. Drop	Cum.	4-cm. Drop	Cum.
0–4	75	75	111	111	7	7	28	28
5–8	60	68	80	95	6	6	23	25
9–12	59	65	85	92	6	6	25	25
13–16	55	62	100	94	6	6	30	26
17–20	60	62			7	6	56	32
21–24	63	62			10	7		
Point of compression	24 cm.		12 cm.		20 cm.		12 cm.	

[a] 0.5 lb. starch causticized with 0.25 lb. caustic per ton solids.

Table 4 shows the effect of increasing the solids from 5 to 10 per cent for both untreated and treated samples.

With a slurry treatment of 0.5 lb. starch and 0.25 lb. caustic per ton of solids, it was necessary to keep the solids down to 5 per cent in order to secure a "settling index" below 10. With slurries of gold ore, it is possible to get excellent results with solids as high as 10 per cent. It appears that the maximum percentage of solids consistent with good operating results depends on the material. The higher the percentage of solids, the shorter is the zone of free settling and in general the better the degree of clarification.

In the following laboratory tests, the percentage of solids was maintained at 5, since in our application to clarification of coal slurries the feed to continuous thickeners is held near 5 per cent solids by using flocculating reagents. Before the use of flocculating reagents the slurry contained as much as 20 per cent solids in the latter part of the week.

Effect of Temperature on Settling Index of Untreated Slurry

The effect of temperature of the slurry on the rate of settling is marked. Table 5 shows the variation of "settling index" with temperature on untreated slurry, but the same variation also holds for treated slurries. For this slurry the rate of settling increases four times in the range from 4° to 40° C. The results are not important practically above 30°, but the rapid decrease in the rate of settling at low temperatures has a direct application in cold weather.

TABLE 5.—*Effect of Temperature on Settling Index of Untreated Slurry Containing 5 Per Cent Solids*

Temperature of Slurry, Deg. C.	Settling Index, Sec. per Cm.	Temperature of Slurry, Deg. C.	Settling Index, Sec. per Cm.
4	146	25	68
10	81	29	63
15	82	35	43
20	78	40	37

Effect of pH of Slurry on Settling Index

The effect of the pH on the rate of settling of solids from a slurry has been well established by other investigators and has been verified in this investigation. Table 6 shows that the minimum rate of settling is observed at a pH of 7.0 and that on either side of this neutral point the rate increases with the most pronounced rise on the alkaline side.

Materials Investigated as Flocculating Agents

At the outset of this investigation, a general study was undertaken of available materials that might be utilized as flocculating agents.

Twenty-five different raw materials were studied, of which 17 of the more interesting are summarized in Table 7, including substances containing starch, protein and cellulose. It was readily apparent that all of these could be used with some degree of success as flocculating agents. Attention need not be confined to starch only in consideration of a suitable material for a flocculating agent. Recent work with proteins and soluble forms of starch has shown clearly that these two general groups of substances can, with proper treatment, produce excellent results. Since starch offers a very attractive possibility as a general flocculating agent and is readily available at a reasonable cost, it is felt advisable to confine the present discussion to the preparation of reagents from this material.

TABLE 6.—*Effect of pH of Slurry on Settling Index*
5.0 PER CENT SOLIDS

pH of Slurry	Settling Index, Sec. per Cm.	pH of Slurry	Settling Index, Sec. per Cm.
Less than 3.0	16	6.4	49
Less than 3.0	21	6.5	53
Less than 3.0	27	6.6	56
4.9	35	7.2	62
6.2	43	10.4	50
6.3	45	10.5	12

TABLE 7.—*Summary of Material Investigated as Flocculating Agents*[a]

Reagent	Settling Index, Sec. per Cm.	Reagent	Settling Index, Sec. per Cm.
Starch "A"	6	Gluten	6
Crude potato starch	5	Zein	11
Refined potato starch	5	Pearl starch "C"	6
Soy bean flour	6	Starch "B"	7
Soy bean meal	7	Powdered starch "D"	6
Wheat bran	6	Beef extract	8
Distillery waste	8	Tapioca flour	6
Crude cornstarch	6	Oak sawdust	14
		Wheat straw	16

[a] 5 per cent solids. Slurry temperature 29° C. Treatment: 0.5 lb. starch causticized with 1.25 per cent NaOH at 100° C. Settling index of untreated slurry, 65 sec. per cm.

METHOD OF PREPARING REAGENTS

The starch reagents were prepared by two methods, causticizing and heating. In causticizing, sufficient water was added to the starch to form a thin paste and caustic solution was added at the desired concentration and temperature. The resultant gel was adequately stirred and then

diluted to a convenient concentration. Since gels containing a higher concentration of starch were difficult to dilute, starch was generally causticized at 5 per cent concentration.

Heat-treated starch solutions prepared below 100° C. were obtained by pouring water heated slightly above the desired temperature into the starch paste. The temperature immediately after addition to the starch paste was taken as the final temperature of preparation. In treatment involving temperatures in excess of 100° C., the starch paste was formed,

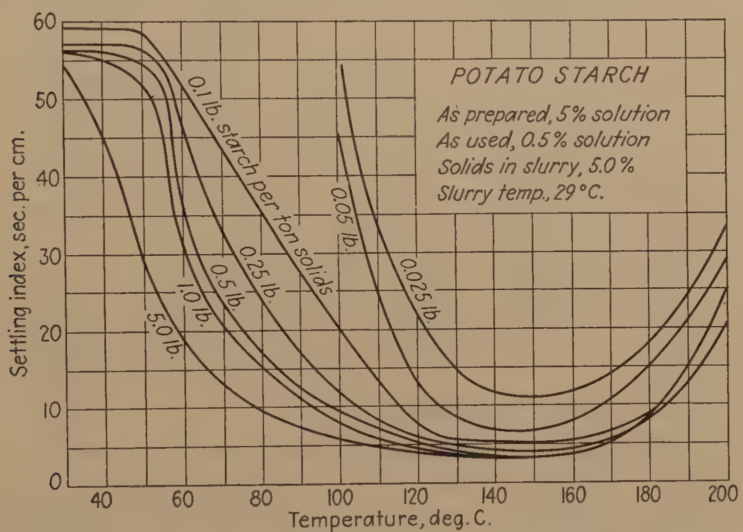

FIG. 1.—EFFECT OF PREPARATION TEMPERATURE ON FLOCCULATING PROPERTIES OF STARCH SOLUTIONS.

diluted to 5 per cent concentration, and the whole was introduced into an autoclave and heated to the desired temperature. In order to facilitate the addition of small amounts of reagent, it was necessary to use solutions diluted to 0.5 per cent starch. It has been found, however, that the reagent may be used at concentrations up to and including 2.5 per cent starch without any substantial decrease in flocculating power. In most cases, the use of reagents containing more than 2.5 per cent starch is impractical because of the difficulty of regulating the rate of addition to the requirements of the plant circuit.

VARIABLES INVOLVED IN PREPARATION OF FLOCCULATING REAGENTS FROM STARCH

Any form of starch treatment that will produce flocculating properties results in the formation of a colloidal solution of at least a portion of the starch. The properties of such solutions are, as a rule, extremely sensitive to the conditions involved in their formation. The following

conditions were found to be important: (1) temperature of preparation of starch solution, (2) concentration of caustic, (3) time of treatment, (4) degree of agitation during treatment, (5) age of reagent.

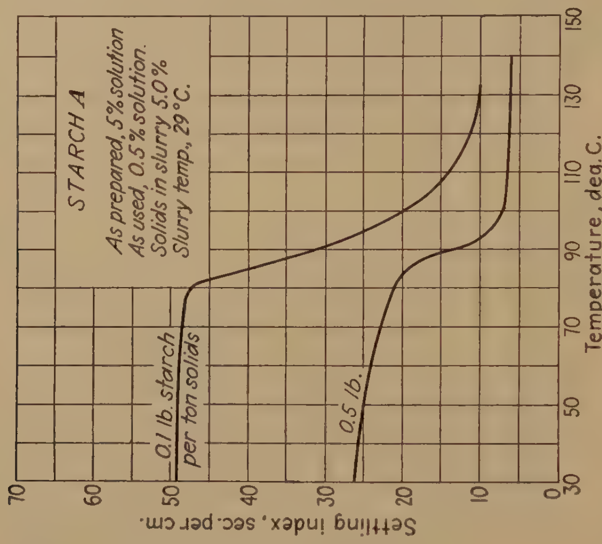

FIG. 3.

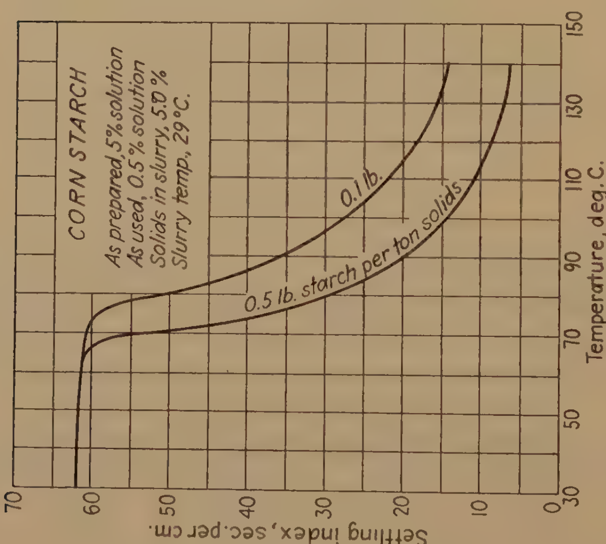

FIG. 2.

FIGS. 2 AND 3.—Effect of preparation temperature on flocculating properties of starch solutions.

Temperature of Preparation of Starch Solution

The importance of temperature in preparation of flocculating reagents from starch cannot be overemphasized. Heating under pressure in the range of 100° to 160° C. will convert starch into a reagent that is as efficient as the product of any form of treatment. The graphs of Figs. 1

to 3 show the change in flocculating properties with temperature for starch from several sources. Examination of a number of starches has shown that these data are, in a general way, applicable to all starches. As the temperature of an aqueous starch paste is raised, the starch grains

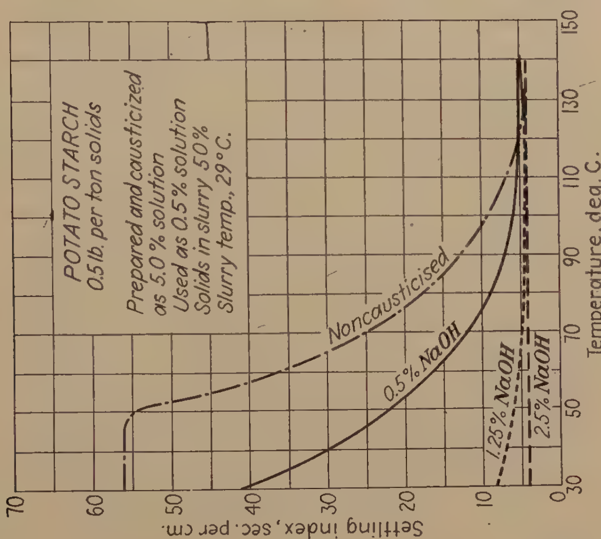

FIG. 5.

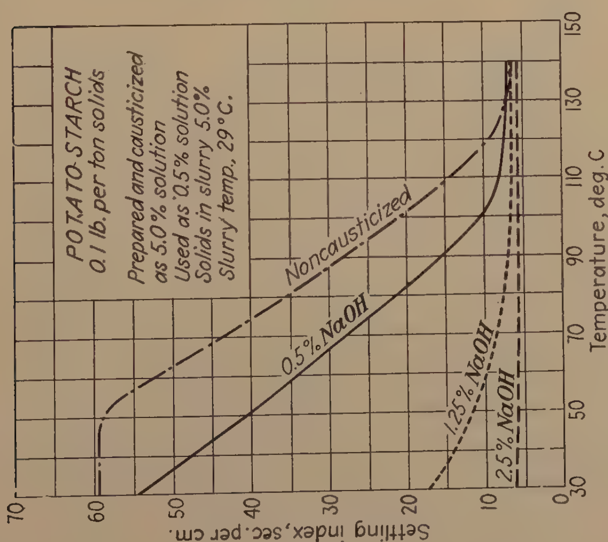

FIG. 4.

FIGS. 4 AND 5.—EFFECT OF PREPARATION TEMPERATURE AND STRENGTH OF CAUSTIC ON FLOCCULATING PROPERTIES OF POTATO STARCH SOLUTIONS.

expand. At temperatures ranging from 55° to 75° C., the starch grains rupture and the soluble portion of the starch is released. Figs. 1 to 3 show clearly that the solution acquires flocculating properties in this temperature range. Only a portion of the starch, however, has been

dispersed at this point and in consequence the solution has not reached its maximum effectiveness. Further increasing the solution temperature above the point at which grain rupture occurs results in a progressive dispersion of this insoluble material and a consequent progressive increase

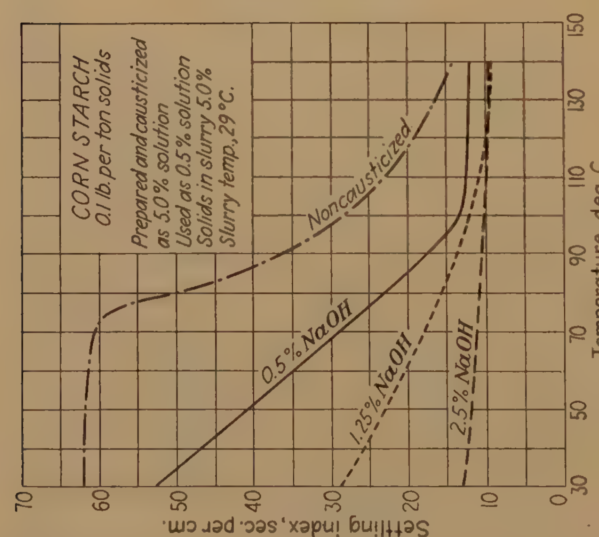

FIG. 6.

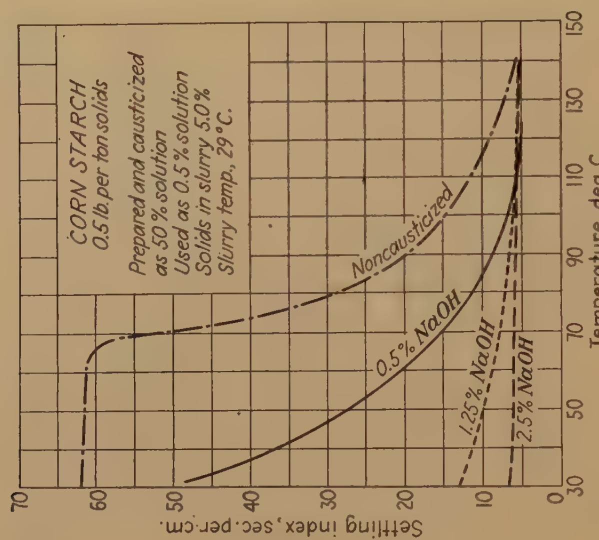

FIG. 7.

FIGS. 6 AND 7.—EFFECT OF PREPARATION TEMPERATURE AND STRENGTH OF CAUSTIC ON FLOCCULATING PROPERTIES OF CORNSTARCH SOLUTIONS.

in the effectiveness of the reagent. At a temperature of approximately 145° C. a starch solution acquires its maximum flocculating power and heating to higher temperature decreases the effectiveness of the starch as a flocculant.

Concentration of Caustic

The term "causticized starch" implies that the starch has been treated with a solution of a caustic reagent. It has been shown previously that

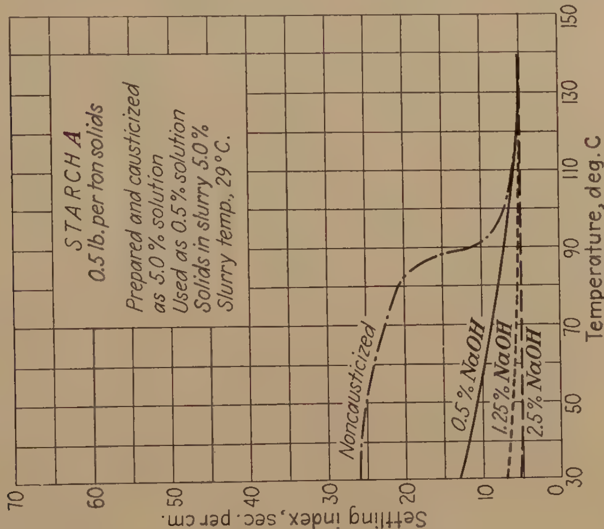

FIG. 9.

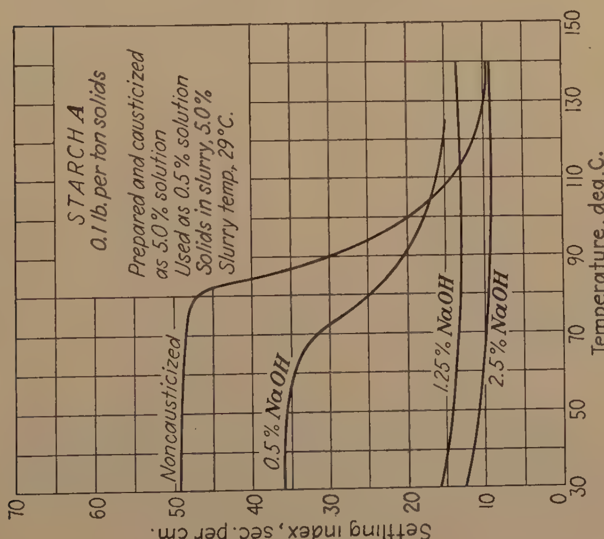

FIG. 8.

FIGS. 8 AND 9.—EFFECT OF PREPARATION TEMPERATURE AND STRENGTH OF CAUSTIC ON FLOCCULATING PROPERTIES OF STARCH A SOLUTIONS

the temperature is an important factor in the preparation of noncausticized solutions. For causticized solutions, two factors must be considered concurrently, temperature and concentration of caustic. Efficient flocculating reagents may be produced at 25° C. by treating the starch with a 2.5 per cent solution of caustic soda. As the strength of caustic

is progressively decreased, it is necessary to increase the temperature in order to produce an equally efficient reagent. Figs. 4 to 9 illustrate this point. The preparation of noncausticized starch reagents at temperatures of 140° to 150° C. (Figs. 1 to 3) produces about the same results as the use of a 2.5 per cent caustic solution at 25° C.

It is important to note that the chief advantage of maintaining optimum conditions in preparation of starch reagents lies in greater efficiency of the reagent. Improperly prepared solutions may produce flocculation if used in sufficiently large quantities, but observations of proper conditions in the preparation of the starch solution make it possible to attain the best results with a minimum amount of starch. Table 8 illustrates the remarkable increase in effectiveness of heat-treated starch solutions with increasing temperatures. An increase of 67° C. in the preparation temperature of the starch solution has decreased by two hundredfold the amount of starch treatment necessary to attain a "settling index" of 10 with this slurry.

TABLE 8.—*Effect of Temperature on Amount of Reagent Necessary to Attain Equivalent Rates of Settling*[a]

Starch, Lb. per Ton Solids[b]	Temperature of Starch Solution, Deg. C.	Settling Index, Sec. per Cm.
5.0	78	10
1.0	92	10
0.5	97	10
0.25	106	10
0.10	117	10
0.05	125	10
0.025	145	10

[a] 5 per cent solids treated with 0.5 per cent potato starch solution. Slurry temperature 29° C.

[b] To convert to parts per million, multiply by 25.

TIME OF TREATMENT

In converting starch to the "soluble form" the treatment must be prolonged for a short period of time in order to obtain the best results. Tables 9 and 10 show the effect of time of heating on the flocculating power of starch reagents. It is apparent that prolonged heating of potato starch below the rupturing temperature has no effect on the starch but that prolonged heating at or above this temperature increases the effectiveness of the starch appreciably. Owing to the time required to cool and dismantle the autoclave, it was impossible to obtain periods of heating of less than 15 min. above 100° C. No improvement was observed when solutions were treated in the autoclave for more than

15 min. In general, 15-min. treatment is to be recommended for both causticized and heat-treated starch solutions.

TABLE 9.—*Effect of Temperature and Time of Heating on Flocculating Properties of Starch Solutions*

Preparation Temperature of Reagent, Deg. C.	Settling Index, Sec. per Cm.				
	Potato Starch			Starch A	
	Time of Heating			Time of Heating	
	0.05 Min.	15 Min.	30 Min.	0.05 Min.	15 Min.
40	60	60		31	20
50	60	60		28	21
60	60	24		27	20
70	37	17		23	21
80	23	12	11	19	9
90	19	8		14	7
100	12	6	6	10	6

TABLE 10.—*Effect of Time of Heating on Flocculating Properties of Potato Starch Solutions*[a]

Time Heating 80° C., Min.	Settling Index, Sec. per Cm.	Time Heating 80° C., Min.	Settling Index, Sec. per Cm.
0.05	23	5.0	13
0.5	18	10.0	13
1.0	17	15.0	12
2.0	15	30.0	11
3.0	14	45.0	11
4.0	13	60.0	11

[a] 0.5 per cent starch solution. 0.5 lb. per ton solids. To convert to parts per million multiply by 25. 5 per cent solids. Slurry temperature, 29° C.

Degree of Agitation during Treatment

Treating a 5 per cent starch paste with a solution of caustic soda yields a thick, viscous gel. It is not sufficient to form this gel and dilute it to the desired concentration; the gel must be thoroughly agitated to produce the best results. Fig. 10 shows the effect of stirring and time of causticizing on the flocculating properties of potato starch. During agitation there is a marked decrease in the rigidity of the gel and when sufficiently agitated the whole is fluid enough to pour freely.

It has been found that a solution of starch should not be treated either by heat or caustic at concentrations in excess of 5 per cent starch. The

difficulty in using stronger solutions is that the gel that is formed cannot readily be dissolved or diluted. Causticizing starch solutions of less than

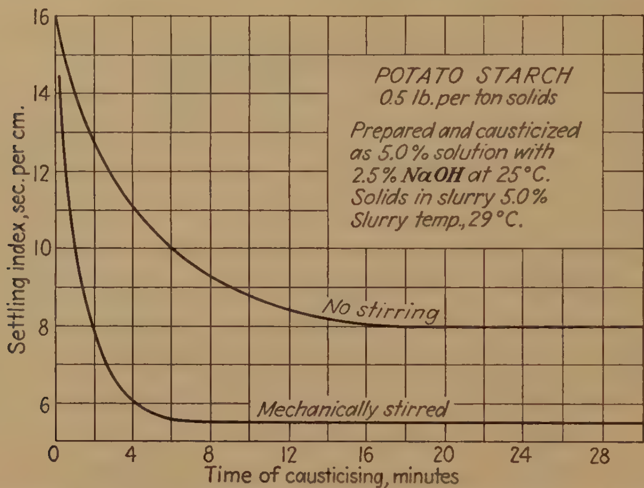

FIG. 10.—EFFECT OF TIME OF STIRRING AND TIME OF CAUSTICIZING ON FLOCCULATING PROPERTIES OF POTATO STARCH.

5 per cent starch give as good results as are obtained by causticizing at 5 per cent and diluting. However, it is cheaper to treat in concentrated

TABLE 11.—*Comparison of Causticizing Crude Potato Starch, Crude Cornstarch, and Starch "A" in Dilute and Concentrated Solutions*[a]

Causticizing Temperature, Deg. C.	Settling Index, Sec. per Cm.											
	Potato Starch				Starch A				Cornstarch			
	0.1 Lb. per Ton[b]		0.5 Lb. per Ton[b]		0.1 Lb. per Ton[b]		0.5 Lb. per Ton[b]		0.1 Lb. per Ton[b]		0.5 Lb. per Ton[b]	
	D	C	D	C	D	C	D	C	D	C	D	C
30	14	17	6	8	16	15	6	7	17	18	11	13
90	8	8	5	5	13	15	6	6	14	19	7	9
100	7	6	5	5	10	12	6	6	10	12	5	6
140	6	6	6	5	15	17	8	6	9	8	6	6

[a] 5 per cent solids. Causticizing solution 1.25 per cent NaOH. Slurry temperature 29° C. C indicates causticized 5 per cent starch, etc.; D, causticized 0.5 per cent starch, etc.

[b] To convert to parts per million multiply by 25.

solutions, for less heat and caustic are required. Table 11 shows the effect of causticizing in dilute and concentrated starch solutions.

Age of Reagent

From a practical point of view, it is extremely important to know how long these flocculating agents will retain their properties. Starch A, prepared by heat-treatment only, deteriorated more rapidly than any other reagent, and after a few days in storage had a dispersing rather than a flocculating action. Other heat-treated starches retained their flocculating properties up to and including three days. Causticized starches retained their flocculating properties for a much longer period of time than did heat-treated starch solutions. Some causticized solutions set for nearly two weeks have shown no appreciable deterioration. The deterioration of these settling reagents is apparently a form of bacteriological fermentation; hence it is reasonable to assume that the reagents would exhibit the properties mentioned above. Starch A contains dextrinized starch, which is readily attacked by bacteria, and it loses some of the flocculating power overnight. The other starches must undergo preliminary transformations before the bacteriological action becomes apparent, so that deterioration is less rapid. Causticized solutions remain unchanged because the alkali inhibits the action of bacteria.

Table 12.—*Effect of Age of Solutions on Flocculating Properties*[a]

Settling Reagent	Temperature of Reagent, Deg. C.	Lb. per Ton Solids[b]		Settling Index, Sec. per Cm.						
				Age Solutions, Days						
		Reagent	Caustic	0	1	2	3	4	5	6
Starch A	80	1.1		6.8	8.4	10.5	47.3	70.7		
Starch A	100	1.1		6.3	7.0	10.7	50.8	56.1		
Causticized starch A	25	0.6	0.6	6.2	6.4	6.6	7.2	8.3	7.6	8.3
Refined potato starch	80	1.1		9.4	8.8	9.1	9.1	11.0	44.0	
Refined potato starch	100	1.1		6.4	6.5	6.7	6.9	12.3	23.3	
Refined potato starch	140	1.1		4.8	4.4	5.0	5.2	6.0	7.0	
Causticized refined potato starch	25	0.6	0.6	6.8	6.3	6.0	5.8	5.8	6.0	
Causticized refined potato starch	60	0.6	0.6	4.5	3.8	4.3	4.2	4.3	4.4	
Crude cornstarch	80	1.1		9.9	9.5	9.8	10.7	29.8	60.2	
Crude cornstarch	100	1.1		6.6	7.3	7.2	6.8	27.5	39.8	
Crude cornstarch	140	1.1		4.6	4.6	4.7	4.9	5.5	7.9	
Causticized crude cornstarch	25	0.6	0.6	9.7	7.3	7.1	7.6	7.0	7.3	
Causticized crude cornstarch	25	0.6	1.1	6.2	6.0	5.5	5.3	6.4	6.9	

[a] 5 per cent solids. Slurry temperature 29° C.

[b] To convert to parts per million multiply by 25.

Suspensions Successfully Flocculated with Starch Solutions

During the investigation the treatment of a number of suspensions other than coal slurry was attempted. Among these were water suspensions of metalliferous ores, sewage, talc, clay, and a slime from a silver electroplating bath. The excellent results attained by various modifications of starch treatment on these suspensions indicate a wide applicability of such treatment. The experience gained in treating these different suspensions has shown clearly that at present there is no general method of starch treatment suitable for all types of suspensions. Lack of space prohibits a complete discussion of these results, but it can be said that proper consideration of the following conditions has made possible the successful treatment of all the suspensions encountered, either by the use of modifications of the starch treatment or by the use of electrolytes in conditioning the slurry for starch treatment:

1. Characteristics of solids
   - *a.* Gravity
   - *b.* Particle size distribution
   - *c.* Surface properties
2. Characteristics of water
   - *a.* Dissolved electrolytes
   - *b.* pH

Table 13 summarizes the more interesting results.

TABLE 13.—*Summary of Results Obtained from Starch Treatment of Various Suspensions*

Type of Suspension	Settling Index, Sec. per Cm.	
	Untreated	With Starch Treatment
Iron ore	18 (incomplete)	2 (complete)
Talc ore	14 (incomplete)	3 (complete)
Gold ore	32 (incomplete)	2 (complete)
Electroplating slime	No appreciable settlement in 6 weeks	5 (complete)
Sewage	300 (incomplete)	9 (complete)

Chemistry of Starch in Relation to Flocculation

Starch occurs in two associated forms: amylopectin and amylose. Amylopectin occurs as the outer skin of the starch grain, while amylose is present as the contents of this envelope. The two differ principally in solubility. Amylose forms a colloidal solution when mixed with water and contains no phosphorus, while amylopectin is normally insoluble in water and contains chemically combined phosphorus. This phosphorus is present as a phosphoric acid ester of starch.

When starch grains are heated in water, the grains expand and rupture at a temperature that varies with starches from different sources. At this temperature, the amylose is released from the confining wall of amylopectin and passes into colloidal solution. Depending on the concentra-

tion of starch, the amylose may be dispersed as a gel or as a solution. Several investigators have shown that the gel-forming properties of starch are due to the amylopectin. It has further been shown by these investigators that the removal of the phosphate radical by hydrolysis and dialysis destroys this gel-forming property.

This provides one explanation for the results obtained in heating and causticizing starches. Table 14 compares the temperature of starch rupture as found with the point at which starch solutions acquire appreciable flocculating properties. The agreement between these figures shows that the first flocculating property acquired by the starch is probably due to the presence of colloidally dispersed amylose.

TABLE 14.—*Comparison of Temperature of Starch Rupture with Temperature at Which Flocculating Properties Appear*

	Rupturing Temperature of Starch, Deg. C.[a]	Temperature at Which Flocculating Properties Appear, Deg. C.
Cornstarch	55.0	56
Potato starch	50.7	52

[a] After Pringsheim.

It has been previously shown that a starch solution has attained only a part of its flocculating properties at the rupturing temperature. It is assumed that the additional flocculating power resulting from higher temperatures is due to changes in the amylopectin. Organic esters are, as a rule, rather easy to hydrolyze either by pressure heating or treatment with acid or alkali. Thus one explanation for this increase in flocculating power would be the hydrolysis of the phosphoric ester of starch (amylopectin) to form soluble amylose and phosphoric acid.

Other investigators have further shown that heating starch at temperatures in excess of 145° C. results in further hydrolysis of the starch into the various decomposition products (dextrins). This provides a ready explanation for the decrease in flocculating power of starch solutions when heated above 145° C.

SUMMARY

Starch solutions capable of flocculating finely divided solids suspended in water can be prepared either by heating under pressure in the range of 100° to 160° C. or by causticizing starch paste. Maximum efficiency with a noncaustic solution is attained when the reagent is prepared at 140° to 145° C. Causticizing temperature depends on the strength of caustic solution used. At 25° C. an efficient reagent can be produced with a 2.5 per cent solution of commercial NaOH. Starch reagent can be prepared most economically by causticizing or heating a 5 per cent starch paste with thorough mixing and diluting. Any starch can be used to

prepare a flocculating reagent, but potato starch is recommended. Solutions prepared by heat alone will retain their properties for three days. Causticized solutions retain their properties for two weeks or longer. The use of these flocculating reagents appears to have a wide range of application.

Acknowledgment

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DISCUSSION

(G. B. Gould presiding)

R. D. Snouffer,* Library, Pa. (written discussion).—Most of my work has been done with a standardized reagent and the amount and the method of treatment varied. The "standard reagent" used was a 1 per cent causticized starch, prepared by rupturing a cold 4 per cent potato-starch suspension with an equal amount of 4 per cent

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sodium hydroxide. The resultant gel was stirred, mechanically, for at least 10 min. and was then diluted with an equal volume of water to give the 1 per cent caustic starch reagent.

In conducting the tests, an instrument was needed to measure relative turbidity, which is a measure of the difference in degrees of flocculation. This was done with the aid of an optical photometer and from these data curves were drawn that showed optimum concentrations and trends with varied treatments.

The vast majority of my work and that of the authors checks very well in the essentials, but I wish to add these words on a few points:

1. The compactness of the thickened slurry is an inverse function of the amount of reagent added. If very little starch is used, or none at all, the rate of clarification is very slow but the settled material is compact and is not readily shaken into suspension. As the amount of starch is increased, the size of the flocs and the percentage of water in the settled product increases, thus allowing the flocs to be readily channeled by fast-moving water or again brought into suspension.

2. With treatment up to the amount that gave the optimum result, there was no sign of starch in the clarified water when tested with an iodine solution. As this is a very sensitive test, it shows that the starch was being completely removed by the coal particles. When amounts of starch were used that were in excess of the optimum range, a blue color due to residual starch was obtained in the water when the iodine was added.

3. No verification of the effect of pH can be made from my investigation, for the caustic-starch reagent apparently worked almost equally well in all waters having pH values ranging from below 6.0 to as high as 13.0.

4. Some plants employ ferric or magnesium hydroxides as flocculants and the method followed is to add reagents such as magnesium or ferric chloride with lime to the water undergoing treatment. As many washery waters contain magnesium or ferric chloride in solution, the only reagent added is the lime. The major inherent difference between methods using the hydroxides and those using starch is that the gelatinous flocculant is formed from true solution in the first case while in the latter it is added to the system. This accounts for the need of pH control with most reagents and the negligible control needed with the starch. However, all the pH values tried lay above the iso-electric point of the starch, which is about 4.6.

5. Often clay can be held in suspension by adding sodium hydroxide, but by introducing a small amount of starch into a basic slip, rapid and complete coagulation of the clay took place. Mixtures of clay and coal seem to flocculate equally well, indicating that separation of the two by preferential flocculation may be difficult.

6. Until near the end of my investigation there was close agreement with the general belief that the starch reagent must be freshly prepared and that it deteriorates with age. Further study showed that the opposite was true, and this substantiates the statement of Gardner and Ray that causticized starch solutions remain unchanged with age.

It was found that when using a freshly prepared 1 per cent causticized starch reagent minimum turbidity was reached on adding 10 to 15 parts per million, as is shown by Fig. 1. Amounts either greater or less gave an increased turbidity, the rise being greatest with a decrease in treatment. A normal curve, Fig. 2, resulting from the use of reagents differing only in age, showed a progressive increase in turbidity, when using equal amounts of treatment. Tests run on reagents from one to fourteen days old showed that the optimum amounts of treatments were less than for a fresh reagent. This is shown in Fig. 3, and indicates that the increase in turbidity when using old solutions was due to an increase in activity of those solutions, and that the progressive rise in turbidity was due to overtreatment rather than to deterioration of the reagent. It is to be noted here that the minimum turbidities obtainable with

the older reagents were slightly higher than those obtained from the freshly prepared solution, but this, however, does not seriously impair their use.

7. When the starch reagent is added to the coal suspension the particles form aggregates or flocs, in which the coal particles are partly surrounded by the starch sol. The charge on the coal particle being negative and that of the emulsoid being positive, there are local shieldings of electronic forces. If a very limited amount of

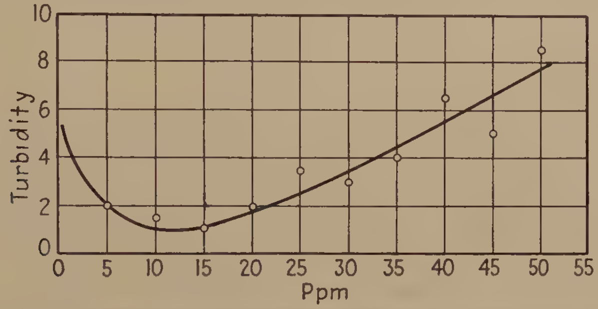

Fig. 11.—Concentration of freshly prepared causticized starch.

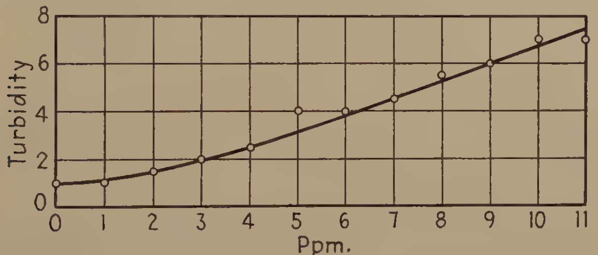

Fig. 12.—Effect of age of reagent at equal concentrations (10 parts per million).

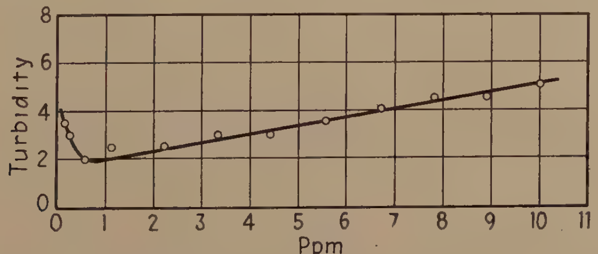

Fig. 13.—Concentration of causticized starch seven days old.

starch is added, the coal is "sensitized," while if more starch reagent is used "coagulation" of the coal takes place. Still higher concentrations of reagent can result in stabilization, if added so as to produce this effect, as was shown by the increase in turbidity by overtreatment.

8. The practice of putting into the flocculating basin all at one time an amount of reagent that has been calculated as satisfactory for treatment in a day's run is not advisable. Such an addition would result in a period of stabilization of the finest material, and as the excess starch was removed from the system an efficient clearing condition would be reached, followed in turn by a period through which the solid content of the system would start to rise.

9. To promote the most satisfactory flocculation, the causticized starch should be continuously added and rapidly agitated for a few seconds, to make sure of complete and uniform distribution throughout the entire suspension. This short period should then be followed by a longer period of gentle mixing so that conditions may favor the gathering together of these particles into large flocs.

D. F. Irvin,* New York, N. Y. (written discussion).—Slime flocculation by use of causticized starch gave such impressive results in the coal-preparation plants of the Pittsburgh Coal Co. that efforts were made to extend its benefits to finely ground pulp of gold-cyanidation plants.

In some such cases, slime conditions exist, which produce slow settling and filtering rates. There is the added condition, absent from coal preparation, wherein valuable gold-cyanide solution must be recovered from the finely divided ore particles.

Tests were made on settling and filtering behavior at a gold-ore cyanide plant where the pulp was notably hard to settle and filter. The best results are believed to typify use of causticized starch as a coagulant on slime from gold-cyanide plants. In general, when using causticized starch, the slime pulp settled much more rapidly than when starch was not used. However, the ultimate pulp density of the slime was somewhat greater when starch was not used, although considerably more time was needed to attain this point. Clarity of the supernatant solution was better when causticized starch was used as a coagulant. A marked difference was observed in the appearance of the thick, settled pulp resulting from use of starch. It was perceptibly "stiffer," and thus showed coagulated effect very plainly. This coagulated pulp gave unusually good filtering performance, as compared to pulp that had received only the ordinary coagulation by lime. The starch-treated pulp gave some increase in dry weight handled per unit filter area; considerable reduction in moisture content of filter cake, and particularly the removal of cake from surface of filter, was much more satisfactory. These tests did not include washing of the cake to remove gold-cyanide solution. A thorough application of this method to cyanidation pulps on a full-scale basis has not yet been arranged, which is regrettable, since it showed promising possibilities for difficult pulps.

Materials that settle and filter easily seldom need starch coagulation.

There has been a general reluctance to make use of any such method on which patent coverage is claimed, as that would require licensing from patentee. This has been a definite hindrance, but such feeling will probably persist, except for some isolated cases where settling and filtering might be very troublesome, and the use of starch as a successful coagulant might justify special arrangements to use the method.

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Measurement of Pressures Developed during the Carbonization of Coal

By Charles C. Russell,* Member A.I.M.E.

(Columbus Meeting, October 1939)

Pressures developed by the coal during the coking process have been responsible for serious trouble to many companies that operate or build by-product coke ovens. The insidious nature of this trouble is indicated by the fact that oven operators, unless they have ascertained the characteristics of the coals they use, cannot determine the existence of dangerous pressures until after the damage has been done. A rather popular misconception is that ease of pushing of the charge at the completion of the coking period indicates that no dangerous pressure has been developed, but this is far from the truth; pressures developed during the coking period are often completely dissipated by the shrinkage of the charge in the completion of coking, and so any criterion based on the ease of pushing is futile. However, methods of testing coals are available that will definitely distinguish the dangerous coals from those that are safe, and such methods have been brought to a high degree of certainty.

The operator should take no risk that involves either a lack of knowledge of the behavior of coals during coking or the failure to believe that certain of the coals or coal mixtures can cause damage to the oven structure. This merely tends to minimize the problem and leads to claims that coking coals of all kinds may be carbonized with no danger of damage to the oven structure provided some particular kind of by-product oven is used. This lack of appreciation of the true situation is in itself one of the main factors leading to damage of coke-oven structures. Coals that develop more than a certain safe pressure cannot be used in by-product coke ovens without causing damage to the side flue walls.

The following paper presents a highly certain method for the measurement of the pressures developed during the carbonization of coal—a method that is the culmination of many years of work on the problem in which a number of methods for the purpose were investigated and used. While actual pressures found may appear to be small when expressed in pounds per square inch, the number of square inches in each oven wall is so large that the total pressure on the oven wall becomes an enormous load.

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Swelling or Expansion

"The coal used . . . will swell during the process of coking to such an extent as to raise the tops from the ovens and push the doors out on each end, though solidly clamped with steel bars . . ."[1] That statement was written in 1906. Korten's[2] paper, published in 1920, is of interest. Many papers were written by German investigators. When Altieri[3] presented his first paper before the American Gas Association in 1935, it aroused very active interest among American coal technologists. In 1938 the interest had grown to such an extent that a symposium on the subject was held at the Production Conference of the American Gas Association.

It is a curious fact that the term "coal expansion" has been widely accepted in the United States as descriptive of the phenomenon in question. This phenomenon, however, concerns the development of pressures within the coal during carbonization that are exerted against the coke-oven walls. An "expanding coal" is considered to be one that will develop sufficient pressure during coking to weaken, distort or otherwise seriously damage the walls. It should be noted that the terms "coal expansion" and "expanding coal" do not carry any implication of pressure development but rather suggest increase in dimensions. Undoubtedly this use of terms has come from laboratory studies of the swelling of coals. Work on this subject has been carried out over a period of years, particularly by English technologists, and has been used principally to determine the relative "coking properties" of various coals and also as a means of separating coals into various ranks. While Brown[4] states that there seems to be no relation between "expansion" and "free swelling," undoubtedly the fact that rapid gas evolution begins during the heating of coal in about the same temperature range as the coal fuses accounts for both of the behaviors studied. Whether a coal "swells" or "expands" during carbonization appears to be a matter of the container in which it is carbonized and the manner in which heat is applied. If there is ample free space, the coal swells up in accordance with its particular characteristics but if the coal is carbonized in a restricted volume, *the effort of resisting the swelling is expressed as a pressure.* The degree of swelling or pressure developed, however, is quite dependent on the fluidity attained during carbonization and also on the rate of evolution of volatile products.[5] These factors are functions of the rank of the coal, its composition, and to some degree the rate at which heat is applied.

In a by-product coke oven, the coal is contained between two vertical walls, usually about 18 in. apart, and the charge in the newer ovens is at least 12 ft. high. Because the ovens are about 40 ft. long, the doors

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[1] References are at the end of the paper.

at either end play a relatively minor role in retaining the coal. With the exception of the coal at the immediate top of the charge, all of the coal within the coke oven is in some measure restricted in volume. The closer to the bottom, the greater is the restriction. It has been shown[6] that granulated material in a narrow bin does not behave like a fluid, and consequently the pressure on any plane below the top of the charge is not equivalent to the weight of the superincumbent coal. However, during carbonization, the effort exerted by the coal in a given position to increase in volume is at least equivalent to the weight of the coal above it.

The degree of packing, expressed as bulk density, has been found by all investigators to be an important function of the amount of swelling or pressure developed. The denser the charge of coal, the greater will be the amount of swelling or pressure developed during carbonization. Auvil and Davis[7] have shown that the increase (or decrease) in volume of a given coal carbonized under specified conditions is directly related to the expansion of the *solid* coal. On this basis they have presented a method of calculation whereby the results obtained at one bulk density can be calculated to any other bulk density.

Test Procedures

Generally the test procedures developed in the United States, and many of those developed in Europe, to determine whether or not a coal will damage coke-oven walls, are in reality modified swelling tests. They are based on the principle that the degree of restriction of coal in the oven can be established in the testing apparatus as well as the bulk density of the charge. If the volume of the coal under test increases when carbonized under these conditions and at an established heating rate, the coal is considered unsafe to use. It was soon found, however, that interpretation of results with respect to large-scale operations was not quite so simple. The variations throughout the rank of coking coals are continuous and no sharp line divides coals into distinct classes. There are many coals that are just on the borderline between being safe and unsafe to use in large-scale ovens according to the test results. Small changes in any of the specified conditions of test place them in one or the other class. Secondly, it was found that the physical state of the coal in the oven is far from being uniform. H. Koppers[8] demonstrated that the bulk density of the coal within the oven is not uniform and may vary over a wide range. This work was confirmed by Koppers Company also (in an unpublished report). Thirdly, the pressure under which the coal is tested is arbitrary, not capable of exact determination in actual practice and not a constant value throughout the oven charge. Lastly, the method of applying heat and rate at which the coal is carbonized in large-scale ovens are very difficult to simulate in test apparatus. Rates

of heating in large-scale ovens can be calculated and the average rate applied to the test apparatus, but, for example, where ovens are purposely operated to produce coke with cool tops such averages do not approach actual oven conditions.

On the other hand, all apparatus used for determining the "expansion" of coal applies heat to only one side. This is of course quite at variance with the conditions in the large-scale ovens where coal is sandwiched between two heating walls. Auvil and Davis[7] have shown that in their test oven, where the coal was heated from one side, the rate of heating drops off very rapidly as the plastic zone approaches the unheated side of the oven. In large-scale practice this unheated side is comparable with the center of the large-scale oven. Even with 5 in. of Sil-O-Cel insulation, on the unheated side, they found some falling off in the heating rate. In large-scale oven practice, these same authors show that there is actually an increase in the rate of heating as the plastic zone approaches the center of the oven.

In addition to all these factors, which are at variance with actual oven conditions, it is difficult if not impossible to make direct comparisons of test results obtained by an empirical method with situations where actual oven damage has occurred. While there has been little in the literature on methods of testing, nothing exists concerning the conditions causing damage to coke ovens. With the lack of opportunity to make correlations between test results and large-scale oven operation, the standardization of an empirical test method for universal use becomes practically impossible.

Study of Behavior of Coals

Notwithstanding these objections to empirical test methods, a large amount of valuable work has been accomplished in the study of the behavior of coal. The work of W. T. Brown,[4] Auvil and Davis,[7] Auvil, Davis and McCartney,[13] and V. J. Altieri[3] are to be especially noted. Brown states that over 900 tests have been made, and his paper presents curves showing the characteristics of hundreds of coals and also a number of demonstrations of the effect of variables on the determination. Brown's work is to be especially commended for the development of apparatus wherein the limit of error in duplicate determinations has been reduced to a minimum. Auvil and Davis have reported especially on the effect of variables on the determination of expansion and have been able to reduce their data to two important laws. Auvil, Davis, and McCartney have very recently determined the effects of heating conditions, bulk density, and coke shrinkage on behavior in two different types of test ovens electrically heated from one side. Altieri's[9] most recent contribution includes not only the development of another type of apparatus but also the demonstration that increasing the load on a given

coal reduces the swelling. After a number of tests have been made at various loads, he extends the curve of the data to show the load required for zero swelling.

Because empirical methods have serious objections, other types of method have been developed. Koppers and Jenkner[10] describe an apparatus for the measurement of pressures developed within the coal charge of a full-scale oven. This consisted of a diaphragm box filled with oil. The box was introduced into the center of the oven charge through a charging hole and it was connected to a manometer by a metal tube. The apparatus was protected from excessive heat, and temperature measurements at the box were made by a thermocouple so that corrections for temperature effect on the pressure could be made. The apparatus had to be withdrawn at the end of the period of water evaporation in the oven or at about one-half the coking period. In one test, pressure up to 0.11 kg. per sq. cm. (1.56 lb. per sq. in.) was obtained, which, the authors say, indicates that the coal must be regarded as highly expanding. The use of the apparatus was discontinued because of the complexity of the procedure.

Ulrich[11] describes an instrument he has designed for the same purpose that apparently has none of the objections that existed in Koppers and Jenkner's apparatus and can be left in the charge throughout the coking period. Results obtained under different conditions are reported in both a test oven and a large-scale oven. The development of this type of apparatus is important in confirming the range of pressures that are found in large-scale ovens, but it should be pointed out that the pressure that is measured is determined at one location in the oven and that the area over which the pressure is measured is an extremely small portion of the area of the oven wall. Determinations of pressure at a number of positions in the oven are necessary in order to obtain a complete survey of the pressures throughout the oven.

Koppers and Jenkner[10] also describe a large test oven in which coal can be carbonized under substantially the same conditions as exist in full-scale ovens. One of the walls of this oven is entirely separate from the rest of the oven, and is supported on wheels. Any movement or pressure developed by the coal in the oven chamber is transmitted to this movable wall. A hydraulic cylinder is attached to the movable wall, so that the pressure of the coal against the wall may be measured by balancing it with the pressure in the cylinder. A large number of determinations were made with this apparatus. One of the principal, and most important, findings thus obtained was that the pressure developed during the carbonization of the coal rises rapidly and reaches a maximum when the two plastic zones meet at the center of the oven. After the peak of pressure is obtained, the pressure drops precipitately because of the shrinkage of coke formed.

This pressure peak is not shown by any other apparatus described in the literature except that described by Ulrich. Certainly no laboratory procedure produces results that show this phenomenon. In some cases this peak is as much as five times that which was obtained during the earlier part of the coking period. It will be shown later that it is impossible to reproduce this behavior in tests where the coal is heated from one side. And it will also be shown that where coal is heated from two sides and volume increase is measured only a very slight indication of this peak can be obtained. The description of this apparatus and the

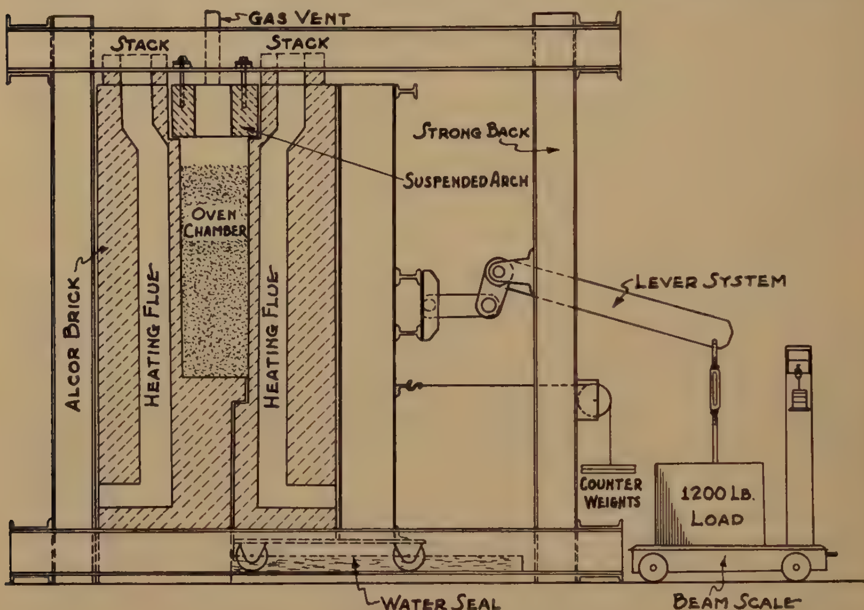

FIG. 1.—OVEN WITH MOVABLE WALL.

results obtained therefrom were published in 1931, and it is strange that this important point should have been either ignored or overlooked in all the published accounts of work on coal expansion since that time.

TEST OVEN WITH MOVABLE WALL

Some time ago Koppers Company designed and erected a test oven based on the principle of that described by Koppers and Jenkner. The general arrangement of this apparatus is shown in Fig. 1. The oven chamber itself has a capacity of about 400 lb. of coal. It is 12 in. wide, 42 in. high and 28 in. long inside the doors. Each wall of the oven has an area of about 1000 sq. in. The heating flues and the face of the oven walls are constructed of standard 9-in. Alcor brick, which has the mechanical properties of silica brick without having the crystallization changes that affect the expansion of silica brick. This brick was selected so that

the oven could be shut down or started up in a relatively short time. The oven is completely insulated with one course of Sil-O-Cel brick and the outside cover is constructed of first-quality firebrick. The roof of the oven is suspended from the top I-beams of the strong back, so that the movable wall will move without contact with the top. The movable wall is constructed on a steel carriage, which is equipped with roller-bearing rollers that rest on 1⅞-in. cold-rolled round steel; the wheels being grooved so that they have the smallest possible contact with the rails to reduce friction. The entire oven sits within a strong back constructed of 8-in. I-beams and channels.

A lever system with a 7:1 ratio mounted on this strong back as indicated in Fig. 1 is used for the transmission of the pressure or movement to the mechanism for measurement. Weights attached to the movable wall through cables as shown counterbalance the weight of these levers. Instead of the hydraulic cylinder used by Koppers and Jenkner for measuring the pressure developed, a simple mechanical principle has been applied, which is far less expensive and without doubt as sensitive as the hydraulic cylinder. It has the additional advantage that the equipment required is available in practically all laboratories. The principle involves the placing of a heavy weight (1200 lb.) on a rigid carriage on a fairly sensitive platform scale. The levers as shown are rigidly connected to this weight through two turnbuckles. Before the test is begun the weight of the load is accurately determined, and then carefully connected to the levers, so that the pull is not more than 20 lb. After the coal is charged into the oven, any pressure developed is transmitted to the wall, thence to the levers and to the load. As the pressure increases, the load on the platform scale is reduced proportionately, and this is determined by weighing the load at frequent intervals. In this way a continuous record of the course of pressure developed during carbonization of a test charge can be obtained. Substantially no movement of the movable wall takes place as long as the pressure in the oven is less than the effort required to lift the weight. What movement occurs is determined by a gauge accurate to 0.001 inch.

The oven has only one door which is lined with 9 in. of brick for insulation. The back of the oven is an integral part of the oven structure and contains a carborundum block 5 in. high having 13 holes spaced 1 in. apart, center to center, that lead into the oven chamber. These are used for the introduction of thermocouples to measure the temperature progression at 1-in. intervals throughout the width of the oven. In order to prevent erroneous temperature readings due to transmission of heat through the steel protecting tubes, the thermocouple wells are staggered in length. The center couple is the longest and each couple proceeding from the center toward each wall is 1½ in. shorter than its adjacent couple.

The oven is heated with coke-oven gas conducted to the flues through open pipe burners. The gas main to the movable wall is of flexible steel tubing, to avoid strain. This also allows the movable wall to be moved out for cleaning. Temperatures of the heating flues are obtained by means of an optical pyrometer at three locations in each wall.

Coal is charged into the oven from a hopper 6 ft. above the top of the oven, through a pipe, into a hole in the center of the suspended top. After charging and leveling, the pipe is removed and a gas vent installed

FIG. 2.—MOVABLE-WALL OVEN ARRANGED TO MEASURE PRESSURE DEVELOPED IN COAL DURING CARBONIZATION.

in its place. The charge is coked until the center thermocouple indicates a temperature of about 600° C. At that temperature the coal has passed through its plastic stage and has become solidified coke. It has also been found that above that temperature no further increase in pressure is obtained. With low-volatile coals that produce high pressures, the maximum pressure has been reached below that temperature and the rapid pressure decrease is well in progress. At the end of the coking period, the coke is raked out of the oven by hand. A photograph of the oven is shown in Fig. 2.

In the first group of tests made with this oven, the wall was allowed to move against a pressure of 2 lb. per sq. in. This pressure was exerted by hanging weights equivalent to that pressure on the lever arms, and

in that case the levers were not connected to the 1200-lb. load described above. The oven was referred to in an earlier paper[12] but few data were at hand when that paper was presented. Fig. 3, curve I, shows the movement of the wall found for a low-volatile Beckley Seam coal. The movement is expressed as percentage of the original oven width. Particular note should be made of the upper end of the curve, where the rate of movement increases rapidly for a 15-min. period, and then drops

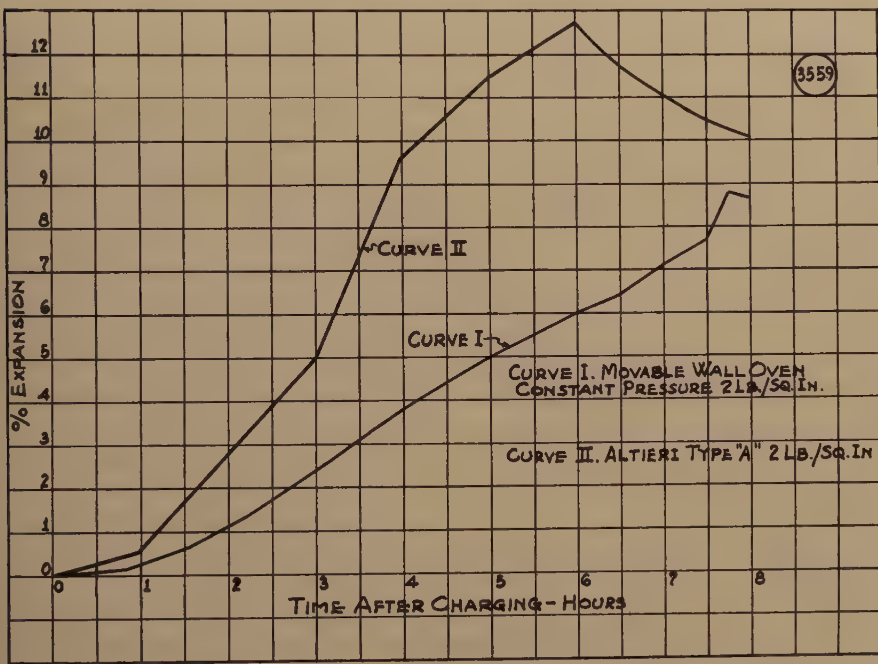

FIG. 3.—EXPANSION TESTS, LOW-VOLATILE COAL FROM BECKLEY SEAM.

off slightly. Curve II shows the data obtained when testing this coal in the Altieri type A apparatus. Data of each are shown in Table 1.

TABLE 1.—*Expansion Tests of Low-volatile Coal from the Beckley Seam*[a]

Oven	Maximum Linear Expansion, Percentage of Original Width	Bulk Density of Coal as Tested, Lb. per Cu. Ft.	Load on Coal, Lb. per Sq. In.	Original Thickness of Coal, In.	Temperature of Heating Wall, End of Test, Deg. C
Koppers movable-wall oven....	8.8	54.0	2	12	971
Altieri type A...............	12.6	55.5	2	6	934

[a] See Fig. 3.

These data indicate that the two methods of testing this coal produce results of comparable magnitude. By calculation of the results from the

Altieri type A to 54.0 lb. per cu. ft., as described by Auvil and Davis,[7] the percentage of maximum expansion is found to be 9.6 per cent, which is within 1 per cent of that found in the movable-wall oven. Sufficient data were not obtained to show that such accuracy can be sustained.

When the movable-wall oven was rearranged to make tests wherein the pressure developed during carbonization was determined, the first coal investigated was also a low-volatile coal from the Beckley Seam

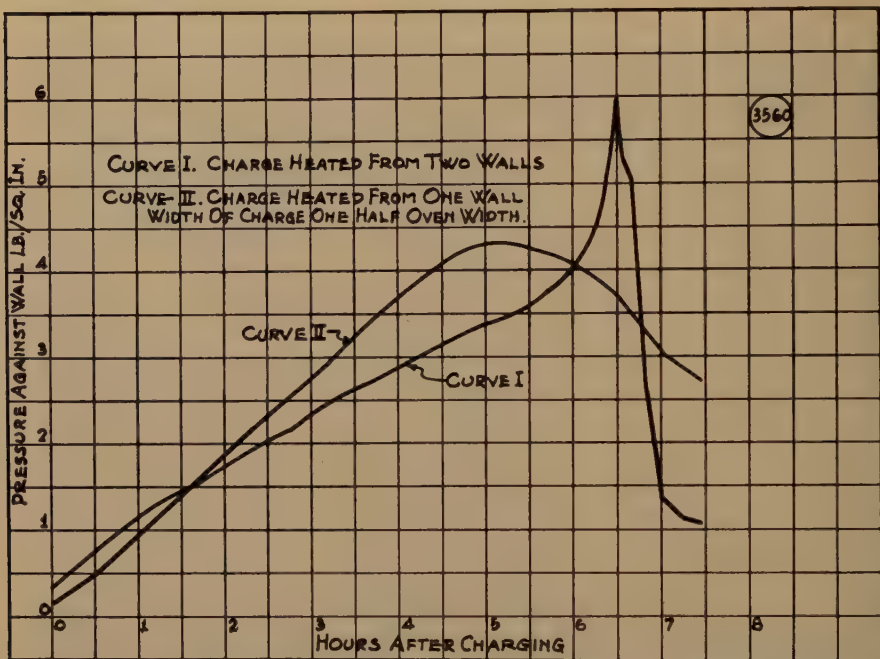

Fig. 4.—Expansion tests, low-volatile coal from Beckley seam, in movable-wall oven.

comparable to that of Table 1. Fig. 4, curve I, shows the data thus obtained. The curve indicates data that are comparable in general with those described by Koppers and Jenkner.[8] Near the end of the coking period a high peak of pressure is found, which occurs at the time of the junction of the two plastic zones.

In order to prove directly that it was the juncture of plastic zones that caused the sudden increase in pressure near the end of the coking period, the oven was arranged so that it was heated from the movable-wall side. By so doing, the coking proceeded from one wall and there was only one plastic zone. To reduce the coking time to be comparable with that of coking from two walls, one-half the oven was filled solidly with brick. This made a coal space one-half that of the normal oven. In order to prevent the cold wall from absorbing heat from the hot wall, the coal space was packed with Sil-O-Cel brick until just before charging

the coal. When the test was ready to be made, the Sil-O-Cel brick were withdrawn, the door put back and luted and the coal charged as soon thereafter as possible. Fig. 4, curve II, shows the data thus obtained. In this test there is no indication of any pressure peak near the end of the coking period. In contrast, the pressure increases at a rate fairly comparable to that in the early part of the coking period, where the coal is heated from two sides, and then more gradually decreases. This curve has much the same general shape as those obtained in test apparatus where the volume change is measured and where the coal is heated from one side.

These tests prove quite conclusively that when two plastic zones are present a high peak of pressure is developed when coking low-volatile coal in an oven heated from two sides. They also indicate that where coal is heated from one side only, giving rise to one plastic zone, no such peak of pressure occurs. The methods of testing now in use in the United States all heat coal from one side only. Under these conditions it is impossible to produce this pressure peak. Furthermore, these same methods all measure expansion of the coal in terms of increase in dimension of the coal charge. Fig. 3, in which is shown the measurement of the increase in width of the movable-wall oven, also shows the increase in rate of change of width that occurs near the end of the coking period. The total change in width during this period of increased rate of change of width amounts to only 0.13 in., or about 13 per cent of the total change in width. On the other hand, the pressure peak is as much as 40 per cent higher than the pressure just prior to the beginning of the rapid rise.

In the case of borderline coals, the knowledge of the existence of this pressure becomes far more important than for coals that are known to be dangerous to coke-oven walls. The present methods of testing use a load of about 2 lb. per sq. in. While this may be an arbitrary value, the interpretation of the test results made on that basis presumes that a coal is dangerous to coke ovens that increases in volume under the load of 2 lb. per sq. in. This literally means that during the carbonization of the coal in the test apparatus pressures greater than 2 lb. per sq. in. are produced. With borderline coals that are tested under that load and heated from one side, pressures of just under 2 lb. per sq. in. may be developed, in which case no increase in volume can occur and the final value obtained may indicate a slight shrinkage. However, because the coal is heated from one side, even the small rapid increase cannot be found near the end of the coking period because of the presence of only one plastic zone. Consequently, the pressure peak of more than 2 lb. per sq. in. that would occur when the coal is coked in a full-scale oven is not detected by the present methods of testing.

Fig. 5 shows an example of the situation described above. Curve I shows the results obtained with the Altieri type A apparatus, and indi-

cates that the coal would be safe to use. In curve II the pressure developed during coking is shown. It will be noted that 1½ hr. after charging the pressure reached 1.75 lb. per sq.in., and that it remained consistently at that value until very near the end of the coking period. At that time the pressure increased rapidly to 4 lb. per sq. in., after which it dropped rapidly. This condition must exist in full-scale ovens, for the walls are rigidly fixed and there is no possibility that a volume change may relieve this pressure. It should also be considered that in this case the pressure

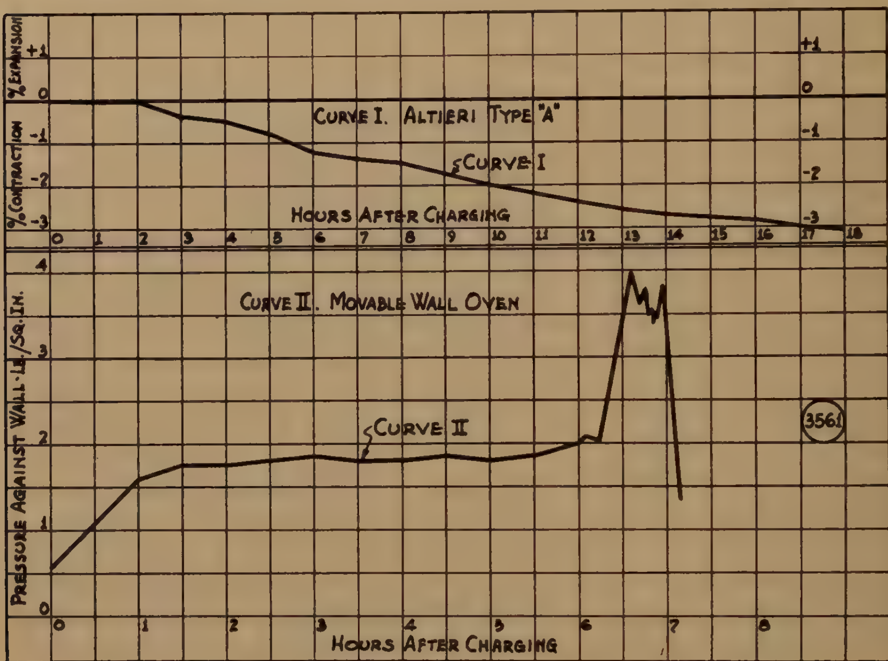

FIG. 5.—COMPARATIVE TESTS OF BORDERLINE COAL MIXTURE.

was developed in a charge only 36 in. high, whereas in full-scale ovens the charge is approximately 144 in. high.

High-volatile coals and various mixtures used regularly in a number of coke-oven plants have been tested with this oven. None of the high-volatile coals have been found to produce a pressure of more than 1 lb. per sq. in. A few mixtures of high-volatile and low-volatile coals that are regularly used produce pressures as high as 1.5 lb. per sq. in. It is also interesting to note that the coals that produce pressures under 1.5 lb. per sq. in. generally do not exhibit a peak of pressure near the end of the coking period similar to that obtained with the low-volatile expanding coals. Probably this can be explained on the basis that these coals become quite fluid, so that even with the juncture of the two plastic zones the fluidity of the coal is high enough to permit the escape of gas without

the creation of high pressures. In the relatively small amount of work that has been done to date on the measurement of fluidity of coal by the Gieseler method, it has been found that the maximum fluidity of a number of low-volatile coals is of the order of 10 units or less, whereas the high-volatile coals are in a range from 1000 to 3000 units. This tends to substantiate the statements just made.

Bulk density of the charge to be tested has been found to be an extremely important factor in affecting the pressure developed. Koppers and Jenkner[10] have pointed this out, and it has been the experience of all investigators, no matter what method was studied. To make the bulk density of the test charge in the movable-wall oven comparable to that found in full-scale ovens is a difficult problem to solve. In the first place, the exact bulk density of coal in the full-scale ovens is difficult to estimate. As mentioned earlier, attempts have been made to determine this in a full-sized oven constructed of wood, but some data are at hand to indicate that the charging of coal into a cold oven is not comparable to charging it into a regular hot coke oven of the same dimensions. The wooden oven method has shown that there is considerable variation in a coal charge. It appears reasonable, however, to make the expansion tests at the highest bulk density that may be presumed to exist in the coke oven. Since bone-dry coal appears to have a fairly uniform bulk density of about 54 lb. per cu. ft., it is believed that that figure should be the minimum value for use in test procedure. To attain this value in the movable-wall oven tests, it has been found that air-dried coal of less than 2 per cent moisture is quite satisfactory.

The description of this movable-wall oven and the presentation of results have been made to demonstrate some of the fundamental factors that must be considered in designing satisfactory testing equipment for the determination of coal expansion. No claim is made that the apparatus described should be adopted as a standard procedure for testing, but any apparatus selected for standard procedure must embody the principles that have been described.

Acknowledgment

Acknowledgment is gratefully made to Mr. C. E. Schaffer, who suggested the mechanical principle used for measurement of the pressures; to Mr. G. V. McGurl, who cooperated in making the tests; to Mr. O. E. Williams, who assisted in some of the earlier work; and to Mr. J. E. Hillemeyer, who prepared the coals and assisted in the operation of the oven.

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DISCUSSION

(*J. E. Tobey presiding*)

J. A. Taylor,* State College, Pa.—The following pressure-time curve (Fig. 6) was obtained in an oven that is heated from only one side. This curve was obtained with a borderline coal, a coal from the Upper Freeport seam with 30 per cent volatile matter. The pressure-scale readings if changed to pounds per square inch are of the

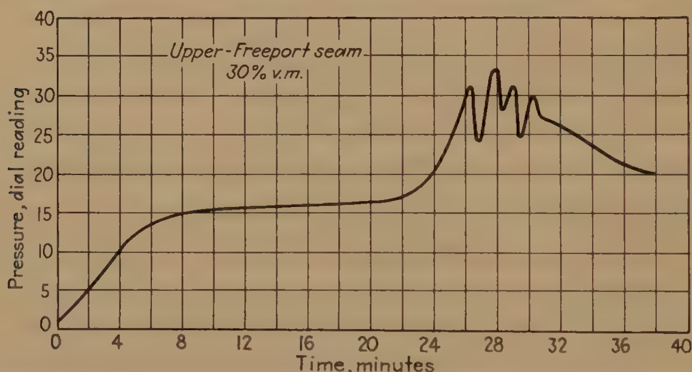

FIG. 6.—PRESSURE-TIME CURVE OBTAINED IN OVEN HEATED FROM ONE SIDE ONLY.

same order of magnitude as in Fig. 5 of the author's paper. It is possible not to get the final peak as shown in this figure with a one-wall-heated oven if care is not taken in the running of such an apparatus; for instance, if the movable wall is allowed to become overheated before the coal is charged into the coking chamber and coking begins from both sides but continues only from one side. In such a case the plastic layer never reaches the wall; it reaches only a layer of semicoke, which is porous and considerably different from the wall or another plastic layer.

From a theoretical viewpoint it seems obvious that the plastic layer is unable to distinguish between the pressure from a solid wall or the pressure from a fluid mass. This viewpoint is upheld by the similarity between the curves in Fig. 5 and Fig. 6.

I agree entirely with Mr. Russell's criticism of expansion measurements that do not give pressure readings.

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* Assistant Professor of Fuel Technology, Pennsylvania State College.

L. A. Shipman,* Knoxville, Tenn.—When the oven is heated from one side only will the plastic layer travel faster than when heated from both sides? Bulk density being the same, does the volatile content of the coal have any relation to pressure developed?

C. C. Russell (author's reply).—The author would require more information than is at hand to discuss the curve presented by Mr. Taylor. It is presumed that the apparatus in which the results were obtained by Mr. Taylor was a small one, for the carbonizing time shown is only 38 min. If the retort were very narrow, as presumably it is, a large proportion of the total width of the coal charge would be in the plastic state. In such a case, the pressures measured would chiefly concern those in the plastic layer, whereas in the large oven described in the paper the pressures measured are the integrated result of the pressure in the plastic layer, the shrinkage of the semicoke and the compression of the unchanged portion of the coal.

When the two plastic layers meet at the center of the oven, the gas that is being evolved within those layers is entrapped in a pasty mass twice as thick as the individual plastic layers. This situation does not occur in an oven that is heated from one side. In the test reported in the paper where the oven was heated from one side, the oven chamber was filled with Sil-O-Cel brick until just before the coal was charged. The maximum temperature of the unheated wall was not more than 200°C.

In answer to Mr. Shipman's questions, the temperature measurements in the test of one-sided heating indicate no more rapid heat travel in the charge than in two-sided heating. The volatile matter of a coal is not related to the pressure developed by that coal except as the volatile matter is related to the plastic properties. There is an apparent relation between volatile matter and pressure but the author believes that the plastic properties are the real criteria that are related to pressure development.

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* Combustion Engineer, Southern Coal and Coke Co.

Production of Low-temperature Coke by the Disco Process

By C. E. Lesher,* Member A.I.M.E.

(New York Meeting, February 1940)

Low-temperature carbonization needs no introduction to the literature on coal. This paper will attempt no review of that literature; it tells the story of the commercial development of one of the processes for carbonizing coal at so-called low temperatures, dealing with the technology and economics of the process rather than with the design of equipment.

The process used is that covered by the patents of C. B. Wisner, with improvements and changes made by the Pittsburgh Coal Carbonization Co. The important and novel feature of the process is the phenomenon of making the product in ball form (Fig. 1). Melting coal dust into smokeless solid fuel in an apparatus as simple as a revolving cylinder of ordinary mild steel has been proved to be practical. Furthermore, it has been found that the process has a wide application, for by applying a knowledge of why and how this balling action is accomplished, the apparatus can be designed and adapted to treat any coal that has a certain minimum agglutinating property.

The operating plant is 20 miles west of Pittsburgh, adjacent to the Champion preparation plant of the Pittsburgh Coal Co. The coal used is cleaned minus 3/8-in. or finer sizes. The products are low-temperature coke, tar and tar products.

Present capacity is 7000 tons per month of low-temperature coke and 140,000 gal. of tar. There is no surplus gas for sale, and none of the light oils are stripped from the gas. The coke is marketed in two sizes, 1 to 2 in. and 1 to 6 in., through retail coal dealers for domestic consumption in hand-fired furnaces, and for fireplace fuel. The smaller size is bagged by dealers. Crude tar and many distillates and road tars are sold.

Acknowledgments

During the 10 years in which the work that is summarized in this paper has been under way, many have contributed time and effort, faithful service and fruitful thinking. Whatever success has attended

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* President, Pittsburgh Coal Carbonization Co., Pittsburgh, Pa.

the efforts to develop and perfect the process and make it operative has been due to those on the daily shifts, who have taken the brunt of the operating troubles. It is in order to name a few who have been on this job from the very first. Mr. J. B. Goode, superintendent, has been in charge of operation, with Mr. C. B. Barmore as first assistant, from the beginning. Mr. A. A. Archer is the engineer who has planned and designed our work from the start. Mr. R. E. Zimmerman, with experience in operation in both the coke and tar plants, has for more than 3 years devoted full time to development research. Mr. Caleb Davies, Jr., is responsible for the refining and marketing of tar and tar products, and conducts the research necessary in the development of uses for low-temperature tar and its derivatives. What is written here does not represent the work of one person. These men are in fact co-authors of this paper. And, of course, had it not been for the constant interest and unwavering support of the Directors of the Pittsburgh Coal Co. this story could not have been written.

History

Interest was turned to low-temperature carbonization as a possible means of profitably disposing of the fine coal that is the result of modern coal preparation and cleaning. The problem thus encountered by this company as early as 1928 has become the problem of a large part of the bituminous coal industry. The modern practice and trend in coal cleaning and preparation for market continues to produce more and more fine coal. These undersizes from minus ½ in. down to minus 48 mesh are largely the resultant of double-screening slack sizes to prepare stoker fuel. In 1937 in the Appalachian and Central Interior high-volatile coal fields there was produced and sent to market a total of 7,415,000 tons of coal that was ⅞ in. or less in size, of which 3,882,000 tons was ⅜ in. or less. The market for such coal is almost entirely for burning as pulverized fuel and the average realization is low. A considerable tonnage of such fine coal is wasted.

Low-temperature carbonization was undertaken by the Pittsburgh Coal Co. because there was no other known method of converting this particular waste material into salable product. In 1929 and 1930 coal was taken to Germany and to Wales for processing. Large tests were made in the Wisner plant at Philo, Ohio, in 1929. An intermittent process of English design was tried. In 1931 a pilot plant using the Wisner process was erected and after 2 years of testing this process was selected for further development. The first commercial-size retort was constructed and put into operation in November 1933. A second retort was put in operation in 1935 and the third early in 1936.

In bringing these three units to their present state of smooth, continuous operation, it was necessary to design apparatus and develop the

process at the same time. Raw fine coal with varying percentages of moisture must be handled; hot fine coal that flows like water and must be kept in motion to prevent firing, must be conveyed, elevated and measured; steel revolving cylinders, 6 to 8 ft. in diameter and 90 to 120 ft. long, must be heated externally with seals for the heating gases as well as for the by-product gas. Handling of raw coal and coke as well as the roasting and carbonizing are continuous, and each step is dependent on the preceding one.

FIG. 1.—LOW-TEMPERATURE COKE BALLS MADE BY DISCO PROCESS.
These balls normally are from 2 to 8 in. in diameter, and are formed in the revolving retort as the coal is carbonized.

Wisner's patents are based on his discovery that certain coals in pulverized form, when heated in air to about 600°F. ("thermodized")* could be resolved into coke of roughly spherical shape ("coal balls") if passed through an externally heated revolving steel retort. At Philo, Ohio, from 1927 to 1929, Wisner employed two inclined revolving retorts, 8 by 30 ft., one above the other. In the upper retort the coal was oxidized and was discharged by gravity to the lower, in which it was carbonized. Each retort was formed of two cylinders, concentric and revolving integrally. Heating gas was forced through the annular space between

* All temperatures are given in degrees Fahrenheit unless otherwise noted.

them. The heating gas was recirculated through a furnace in which its temperature was raised to the desired point. By-product gas was exhausted from the upper end of the lower retort and the solid product was withdrawn from the lower end through a sealed discharging mechanism. Low-temperature coke balls were produced with this apparatus but the longest period that the apparatus could be made to function was 102 hr., because of mechanical as well as process imperfections. The pilot plant of the Pittsburgh Coal Carbonization Co., built in 1931, followed the design of the plant at Philo.

Operation at Champion Plant

The Disco plant of the Pittsburgh Coal Carbonization Co. is a self-contained operating unit. Coal from the Champion No. 1 cleaning plant of the Pittsburgh Coal Co. is minus 8-mesh or minus 3/8-in., and has been cleaned. Low-temperature coke is air-cooled and screened. Tar is collected and delivered to the storage tanks of the adjacent tar plant. By-product coal gas is used raw for heating the roasters and carbonizers. All coke that passes through a 1-in. screen is crushed to minus 1/4 in. and recirculated in the carbonizers.

The process is continuous and the plant is operated 24 hr. a day and 7 days a week. Because the coke is a household furnace fuel the market is seasonal and during the summer months a part of the production is stocked. Present practice is to shut down the plant twice a year for overhauling of conveying, screening and handling equipment. Individual units may be taken off production at any time for cleaning scale from the retort. Unit 3 was operated for 200 days with but 25 min. lost time from mid-September 1938 to the end of March 1939, and for 164 days with 1 hr. 40 min. lost time from May 16 to Oct. 26, 1939. Such continuous operating records are the result of good design and careful maintenance.

Raw-coal handling[1] is by belts and conveyors by which the feed is distributed to 24-hr. storage bins over each of the three retorts. A disk feeder under each bin delivers a uniform supply of coal to the unit. General dimensions of the three units are given in Table 1 and the flow diagram in Fig. 2 illustrates present design for a strongly coking coal such as is used at Champion.

Heating

Heating is accomplished in a closed circuit with high-speed gas. In a refractory-lined furnace by-product gas in controlled amounts is burned and mixed with recirculated gas so as to maintain a temperature of about 1020° in the mixture. The heating gas is forced through the annular

[1] References are at the end of the paper.

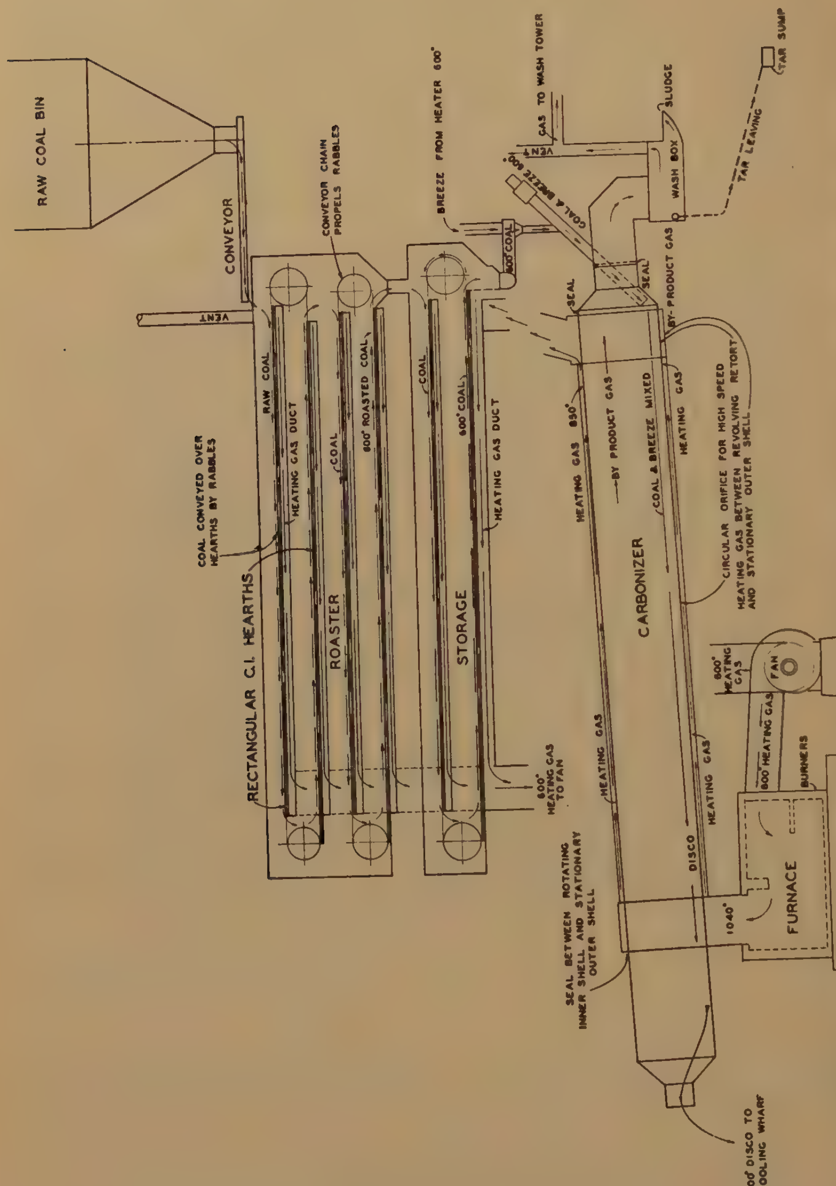


FIG. 2.—FLOW DIAGRAM OF LOW-TEMPERATURE COKING UNIT, USING DISCO PROCESS, FOR STRONGLY COKING COAL.

space around the carbonizing retort at a speed of about 3500 ft. per minute. Its temperature as it leaves the carbonizer is between 850° and 900°. The roaster and storage conveyor are heated by the gas after it leaves the carbonizer. There is a drop in temperature of about 200° in this part of the circuit, and the gas reaches the inlet of the recirculating

TABLE 1.—*Data on Carbonizers, Disco Plant, Champion, Pa.*

Item	Unit No. 1	Unit No. 2	Unit No. 3
Diameter	6′ 0″	8′ 0″	8′ 0″
Length over all, cold	91′ 3½″	106′ 10″	126′ 10″
Center line tires (cold)	54′ 0″	84′ 10″	104′ 6″
Slope	⅛″ in 12″	⅜″ in 12″	½″ in 12″
Total rotating weight, lb	110,000	106,900	138,000
Motor, hp	30	30	50
Speed, r.p.m	900	720	900
Horsepower used	19–21	16	21–23
Carbonizer, r.p.m	5.25	3.44	2.89
Maximum expansion, in	3½	5⅞	6⅞
Center line hoods	76′ 0″	71′ 3″	90′ 9″
High-speed heat area, sq. ft	1,432	1,791	2,306
Feed raw coal per hour, lb	6,250	9,580	12,330
Feed breeze per hour, lb	1,875	2,875	3,700
Total feed per hour, lb	8,125	12,455	16,030
High-speed heating area, lb. per hr. per sq. ft	5.67	6.95	6.95
Heating gas, lb. per min	825	1,059	1,307
Feed, lb. per lb. heating gas	0.164	0.196	0.205
Heating gas at 700°, cu. ft. per min	25,000	32,000	39,500
Combustion air blower, cu. ft. per min	1,680	2,300	2,300
British thermal units per hour	4,356,000	5,592,000	6,904,000
Tons coal per 24 hr	75	115	148
Size heating gas fan, in	42 × 12	48 × 15	54 × 18
Fan motor, hp	25	25	20–30
Speed, r.p.m	720	600	450[a]
Horsepower used	20	21	28
Roaster-hearth area, sq. ft	1,200	1,955	1,955
Storage capacity, hours	1–3	1–3	1–3

[a] Rewound 30-hp. 600-r.p.m. motor.

fan at between 650° and 700°. Between the fan outlet and the furnace the excess gas is vented. The products of combustion do not contact the coal in process.

The quantity of heat required to carbonize a pound of coal depends on the particular coal and the final temperature of carbonization. At

Champion, the final temperature of all products—coke, tar, gas and water vapor—is 850°. The sensible heat in material leaving the system is then 850°–70°, or 780° × 0.3, or 234 B.t.u. per pound of dry feed, where 0.3 is the specific heat of products. To this must be added 10 B.t.u. for each per cent of moisture in the feed.

In the three units at Champion there are produced 50,000 cu. ft. of by-product gas per hour, with gross heat value of 414 B.t.u. and net of 378 B.t.u. per cu. ft. Eighty-five per cent of this gas is used in the process, in addition to which pilot burners using natural gas add 500,000 B.t.u. per hour. The heat input from gas burners is 85 per cent of 50,000 cu. ft. at 378 B.t.u., or 16,065,000 plus 500,000 B.t.u., a total of 16,565,000 B.t.u. per hour. Coal carbonized is 325 tons per day, or 27,000 lb. per hour. There are therefore 698 B.t.u. in the by-product gas from each pound of coal carbonized. Breeze in the feed is 28 per cent of the coal. Heat used is 612 B.t.u. per pound of coal processed, and 478 B.t.u. per pound of coal and breeze in the plant feed.

In the roasting operation the theoretical heat required to heat the coal to 600° is, assuming 5 per cent moisture in the coal, 191 B.t.u. per pound. Because the roasting is in part an oxidation that supplies heat, this theoretical quantity of heat is not required. In practice, the roaster drys the coal and above 300° supplies heat internally in undetermined amount. One test of gases vented from a roaster showed 6 cu. ft. of gas per pound of coal, in which there was 18.5 per cent O_2, 0.2 per cent CO_2 and 0.3 per cent CO.

In the carbonizer the heat required may be estimated at 75 B.t.u. per pound of feed, since the increase in temperature is from 600° to 850° and all products are assumed to have a specific heat of 0.3. The temperature drop in heating gas in its passage along the retort is from 1020° to 900°, or 29.5 B.t.u. per pound of gas. Five pounds of heating gas are circulated for each pound of coal and breeze in process. The heat supplied by the circulating gas is then 147.5 B.t.u. per pound of feed.

TABLE 2.—*Approximate Heat Balance*

Input	B.t.u. per Pound of Total Feed	Distribution	B.t.u. per Pound of Total Feed
From combustion of gases in furnace	478	Sensible heat in products (43.5 per cent)	234
From "open-end" operation	60	Vented (32.5 per cent)	175
		Losses, through radiation and seals (24.0 per cent)	129
	538		538

There are two other sources of heat for the coal and breeze in process in the retort. One is the exothermic reaction in the coal at the tempera-

ture of carbonization and the other is the effect of "open-end" operation (see page 342). The amount of heat from the exothermic reaction of coking this coal has been estimated[2] at 53 B.t.u. per pound of coal. In this study there has been no possible means of checking this figure.

By "open-end" operation, air and steam are drawn into the discharge opening of the carbonizer and by combining with hot products in the retort generate heat. By two separate methods of test and calculation, the quantity of heat so produced directly inside the carbonizing retort have been estimated at 60 B.t.u. per pound of feed.

An approximate heat balance is given in Table 2.

Coal Treatment

Roasters.—The flow of coal is from the bin to the roaster. Here in thin layers on horizontal cast-iron hearths the coal is stirred and conveyed by rabbles for about 2 hr. Oxidation is controlled by the time and temperature of treatment and the quantity of air admitted to the roaster. Two designs of roaster are used at Champion. One has long, narrow hearths over which the rabble arms are propelled by conveyor chains; the other is the circular, multiple-hearth type, differing from standard design in that the heating gases are in closed ducts and do not come into contact with the material in process. Roasting the coal to oxidize it partially is necessary for highly coking coals, to reduce the agglutinating property before carbonizing in the revolving retort.

Storage Conveyor.—Coal from the roaster, at about 600°, is conveyed to the storage conveyor, which is a reservoir in which the hot coal is kept in motion and is recirculated and mixed, which holds a supply sufficient for from 1 to 3 hr. for the carbonizer. The storage conveyor serves to smooth out changes in coal quality as well as to equalize irregularities in the flow from the roaster. A feeder measures a constant volume from the storage for the feed to the carbonizer. Two types of apparatus are used as storage conveyor. One has two rectangular hearths over which the roasted coal is carried continuously around by buckets attached to a chain; the other is a double revolving inclined tube. Coal enters the inner tube, passes down its length, and is returned by a spiral that fills the space between the tubes. Means are provided for recirculation within the cylinder. A feeder takes out the required amount for the carbonizer and returns the remainder to the central tube. Coal in the storage conveyor is maintained at 600° by a portion of the heating gas in ducts. No air is admitted to the tubular storage systems.

Carbonizer.—The carbonizers are inclined rotating steel cylinders. Surrounding the rotary cylinder is a stationary shell insulated on the outside. Heating gases pass through the annular space between the two shells at high speed and heat the revolving element. Into the upper end of the revolving retort coal from the storage conveyor is carried by a

screw and by-product gas is withdrawn by an exhauster. Mixed with the coal in the feed screw is a definite proportion of minus ¼-in. preheated coke breeze. This mixture is carried down the inclined retort and carbonized into coke balls during its progress. The lower end of the retort is open to the atmosphere, and the hot low-temperature coke is discharged in a continuous stream into a gathering conveyor.

Cooling the Coke

Coke that will pass a 2-in. screen is water-quenched, but it is necessary to cool the larger low-temperature coke balls in air, as quenching with water causes shrinkage cracks that destroy the structure of the balls. When discharged from the carbonizer at 850° the coke balls are so reactive that they will immediately take fire unless kept in motion. Movement of the balls, slight and frequent, is accomplished on a cooling wharf with mechanical agitation. The wharf is made up of slotted, hinged, cast-iron grates in step formation, the whole inclined at an angle of 18°. Trippers in an endless chain raise and lower each grate in turn, imparting a wave-like motion to the bed of coke, and moving it slowly down the wharf. After 2½ hr. of air-cooling the temperature of the larger (6-in.) balls has been reduced to less than 400°. At this temperature the coke can be screened and loaded into cars. The rate of cooling of coke balls of different sizes in still air is shown in Fig. 3.

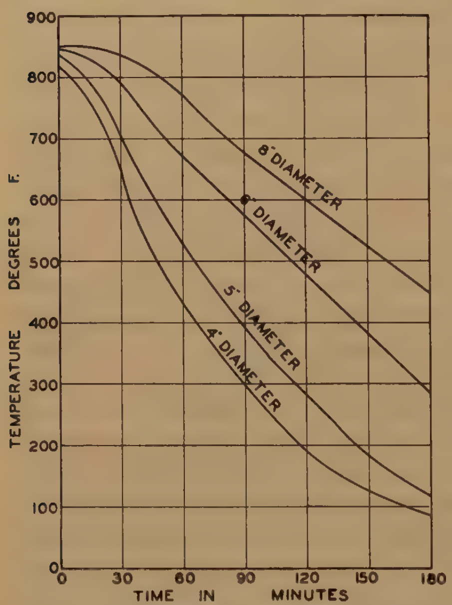


Fig. 3.—Rates of cooling of low-temperature coke balls of different size, in still air.

Breeze

Breeze, the undersize from production, is recirculated. Normally the products of the carbonizers are coke balls and fine material resulting from the grinding of the balls in the lower end of the retort. These fines are screened out through ½-in. perforations in the discharge nozzle of each retort. Breakage through 1-in. screens from handling while cooling and screening the product, together with these fines, is passed through rolls to reduce oversize to minus ¼ in. and returned to bins. From these bins

definite amounts are withdrawn and after preheating (in tubes in the heating-gas circuits) mixed with the hot coal in the feed to the carbonizers.

In the current operations of this plant screening at 1 in. there is no excess breeze, but instead a deficiency for good operation of about 2 per cent of coal feed. Degradation from stocking supplies this deficiency.

Hydrocarbon Gas

By-product gas is withdrawn from the upper end of the carbonizers. In a stationary insulated chamber, dust is settled out. From this point the handling of gas follows regular practice. Tar is condensed by direct water sprays and an exhauster and small gas holder collect the raw gas and deliver it to the burners at each furnace.

Properties of Disco

In physical appearance Disco is dense grained and black. Its shape is characteristic of the process, irregularly rounded balls, or fragments of balls. The shatter index is above 70 per cent retained on a 2-in. screen, Chemical analyses and physical properties of Disco are given in Table 3. which also shows the analysis of the coal from which the coke is made. Table 4 gives screen analyses of the coals used and the breeze recirculated.

Table 3.—*Analyses of Minus 8-mesh Feed Coal and Disco (dry basis), Disco Plant*

Sample	Proximate Analysis, Per Cent			Sulphur	B.t.u.	Fusion of Ash
	Ash	Volatile Matter	Fixed Carbon			
Coal[a]	8.0	37.5	54.5	2.20	13,710	2150
Disco[b,c,d]	10.2	17.0	72.8	2.10	13,100	2150

[a] Agglutinating value minus 8-mesh coal is 11.5 (15:1).

[b] Disco apparent specific gravity is 0.856; true specific gravity, 1.456; percentage of porosity, 41.2.

[c] Disco shatter index, average of daily shift samples for last six months, is 71.5 per cent retained on 2-in. screen.

[d] Disco weight per cubic foot is 33.2 lb. for plus 1-in. size.

Reactivity.—Tests made by the Coal Research Laboratory of the Carnegie Institute of Technology, Pittsburgh, Pa., according to a private communication from Dr. H. H. Lowry, show Disco made at Champion No. 1 to be highly reactive; in fact, even more reactive than the Pittsburgh-seam coal (Edenborn mine) itself. They show values for T_{15}, deg. C. of 205 and T_{75}, deg. C. of 260 for Disco, as, for example, a T_{15} of 230 and T_{75} of 285 for the Edenborn coal, and a T_{15} of 415 and T_{75} of 495 for ordinary high-temperature by-product coke. Values T_{15} and T_{75} are

the temperatures at which the rate of heat release by oxidation of a sized sample becomes great enough to raise its temperature at rates of 15°C. per min. and 75°C. per min., respectively. The lower the temperatures are, the more reactive the fuel; and the less the numerical difference between T_{75} and T_{15} the faster the rate of reaction increases with temperature. Their method for determining reactivity has been published.[3]

TABLE 4.—*Screen Analyses of Disco-plant Feed*[a]
TYLER STANDARD MESH

Size	Minus 8 – Mesh Coal		3/8 × 0 Coal		Recirculating Breeze	
	Per Cent	Cumulative Per Cent	Per Cent	Cumulative Per Cent	Per Cent	Cumulative Per Cent
+ 4-mesh	1.5	1.5	29.0	29.0	18.0	18.0
4 to 8	6.5	8.0	22.0	51.0	26.0	44.0
8 to 14	25.7	33.7	15.0	66.0	19.5	63.5
14 to 48	42.0	75.7	18.5	84.5	24.0	87.5
48 to 100	10.2	85.9	6.5	91.0	6.0	93.5
100 to 200	5.7	91.6	4.0	95.0	3.5	97.0
– 200	8.4	100.0	5.0	100.0	3.0	100.0
	100.0		100.0		100.0	

[a] Plant is usually fed the minus 8-mesh coal, which is the aspirated fines from the washed 3/8 to 0 coal. Occasionally straight 3/8 to 0 is used.

It is interesting that although there are numerous methods for determining reactivity of a coal or coke, Disco shows itself to be highly reactive regardless of the method used. Yancey reports,[4] according to a modified Kreulen's method, an ignition temperature for Disco of 375°C., as low if not lower than many coals. By-product coke by this method will have ignition points varying from 460° to 520°C. Reactivity of Disco by Kreulen's method at 600°C. shows 103 milligrams of CO_2 formed per square centimeter in 5 min., as against 28 to 75 for various by-product cokes. The modified Kreulen method consists in weighing the amount of carbon dioxide produced in 5 min. by passing air over the surface of finely divided coke at a temperature of 600°C.

BATCH RETORT

For testing coals and for research on the Disco process, an intermittent or batch retort is used. This is a revolving cylinder, externally heated, in which a batch of coal can be given the heat-treatment in a revolving shell corresponding to that in a continuous plant. Two sizes are in use, one 3 ft. in diameter and 4½ ft. long; the other 10 ft. in diameter and 6 ft.

long. Through a hollow shaft, air and steam may be forced into the retort, and distillation vapors removed. The coal is weighed and charged into the cylinder through a door when the apparatus is cold, or through a 3-in. opening closed by a plug when it is hot. The final product is removed through the door. By-product gas can be collected and measured and the tar and liquor condensed. Temperatures of heating gas and of the steel shell are measured by thermocouples. Every reaction of the continuous retort is duplicated in the batch machine. In this machine many coals are tested and data obtained for design of large retorts. This machine has been very useful in studying the cycle of coking and ball formation. Fig. 4 is a typical data sheet giving the time and temperature curves for a test.

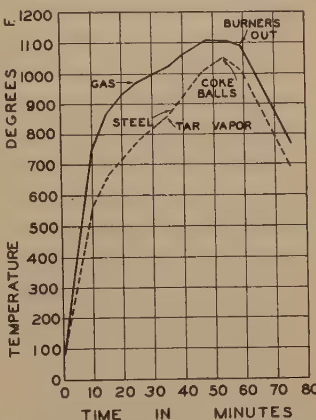


Fig. 4.—Time and temperature curves for 10-foot diameter batch machine test, making Disco from minus 8-mesh Champion No. 1 coal. Test made September 28, 1939.

Charge: 2000 lb. minus 8-mesh coal, pretreated for 2 hr. in Unit 1 roaster of Disco plant, and 500 lb. of minus 4-mesh breeze; total 2500 lb.

Retort speed, 4.7 r.p.m. Coal at 600° charged into cold machine. Curves begin after charging was completed and retort closed. Initial tar vapor appeared in 35 minutes after start and when steel shell temperature was 870°. Coke balls were made in 53 minutes after start with steel temperature at 1060°. Burners were off 58 minutes after start and retort was opened and product taken out 75 minutes after start. Product was well rounded, dense, coke balls.

The Disco Process

The continuous heating and carbonizing of coal in an inclined revolving retort to form low-temperature coke in rounded, homogeneous ball-shaped pieces is the essence of the process.

The conclusion was reached early in this work that the reactions taking place in the oxidation and carbonization of coal at low temperatures are sensitive and that successful operation depended on the development of suitable and adequate control of these reactions. But the reactions must be known to be controlled, and what takes place in a revolving steel retort in operation is impossible of direct observation. Observations were made of the interior of the retorts after operation for varying periods of time. The character of the material found and the crusts and scales adhering to the steel gave valuable clues to the reactions by which the coke balls are made.

Wisner[5,6] ascribed the formation of "coal balls" to the differential heating of the coal next to the revolving shell of the retort and the tumbling action of the retort. Roberts[7] believes that "spheroids of semi-coke are formed in snowball fashion." Our explanation of what takes place in the formation of these coke balls can be stated as follows:

The mixture of preheated, finely divided coal and breeze is conveyed by the revolving action of the inclined retort toward the discharge end. At the point where the incoming material meets steel the temperature of the shell is about 800° and the temperature of the coal mixture is increased as it progresses until it attains the temperature at which it becomes soft or plastic. Yellowish hydrocarbon vapor is given off as the coal softens. In the dry state the feed does not stick to the steel. The hot, dry, fine coal and breeze flow as freely as a fluid. At a temperature above 700° the coal softens, gives off hydrocarbon vapors, and distillation, as generally understood, has begun. The exact temperature at which coal softens is different for different coals, but the range is generally between 700° and 800°. Distillation really begins at lower temperatures. Some hydrocarbon vapors are given off at temperatures below 600°. In addition to H_2O, CO_2 and CO, there are small amounts of light, volatile hydrocarbons, usually detectable by a white oily vapor as against the yellowish vapor that develops at or near the softening point of the coal.

As the coal softens it becomes sticky. The whole mass does not melt into a fluid state, but some constituents of the coal soften, wet the surfaces of adjacent or inert particles and bind the whole into a loosely coherent mass. In this condition its coefficient of friction on steel is greatly increased. Instead of a thin layer, loosely flowing, there is a thickening of the bed, because the increase in friction not only retards its flow down the retort but increases the length of climb up the arc of the revolving shell. The charge has changed from dry to wet and is now in the plastic stage. This transition takes place within a short interval.

Revolving action of the retort raises the mass and as it falls over it is kneaded and disintegrated into smaller masses, quite irregular in shape and size. The loosely coherent mass literally falls apart. The segregation into small masses that later become hardened as coke balls is not dependent on "snowballing" action—that is, the larger masses are not the result of small pieces rolling down a surface of material and increasing in size as a snowball gains in size as it rolls down a slope. The size of the masses that finally merge as coke balls is determined by the agglutinating property of the coal at the time it reaches its softening temperature in the retort.

Transition through the plastic stage is rapid, about 5 to 10 min., perhaps even less for some coals. The semifluid mixture is torn apart by the revolving action of the retort into soft masses that quickly attain sufficient consistency to be individual pieces. Only under most unusual conditions do two such pieces coalesce, or is one ball rolled within another. Free to move along and out of the way of oncoming coal as in the inclined revolving retort with continuous feed, these smaller masses of soft material become dry and solid, with roughly spherical shape.

Size and structure of the coke balls are the important objectives in control of the process. The desired size of product is a range between 1 and 8 in. The structure should be dense, homogeneous and resistant to shatter. Ash, sulphur and fusion point of ash are determined by the coal used. The content of volatile matter depends on the treatment in the retort. It has been found that a product with satisfactory structure will have a volatile matter content of about 16 per cent. Experience has shown that for any given coal:

1. The *size* of the pieces that emerge as low-temperature coke balls is determined by the agglutinating property of the coal at the time it reaches its softening temperature in the retort.

2. The *structure* of the low-temperature coke is determined: (*a*) by the character and proportion of inerts mixed with the coal and (*b*) by the rate and maximum temperature of heating and the effective control of the rates and temperatures in preplastic, plastic and postplastic ranges.

Influence of Plasticity

Dry coal melts to form coke balls because it becomes plastic when heated. Coal plasticity is described by J. D. Davis:[8]

> Heating coal in the absence of air causes part of its constituents to melt and other parts to produce liquids by decomposition. When these liquids are present in large enough amounts they disperse the solid colloidal particles of coal to form an organophilic sol commonly called the plastic state. Extended preheating (or oxidation) in the preplastic state decomposes the liquid forming material to such an extent that within the plastic range there may no longer be left a sufficient quantity in which to disperse the solids. A high rate of heating through the preplastic range increases the amount of fluid material causing solvation and therefore some coals form a much better coke when heated at a high rate than when coked at normal rates. The formation of the colloid sol called the plastic state is a delicate process and depends greatly on the thorough dispersion of the constituents of the coal.

Experience with the Disco process has definitely established the fact that size of ball is determined by the plastic property of the material in process at the time it is at its softening temperature. A measure of this property is the agglutinating index.[9] Strongly coking coals have a high index and noncoking coals have no measurable agglutinating property. A coal of moderate coking property will make coke balls of desired size with little or no preliminary treatment. A weakly coking coal must be heated rapidly through the preplastic range if coke balls are to be produced.

Bituminous material with high agglutinating index, such as coal-tar pitch, or a strongly coking coal can be mixed with weakly coking coal, or even inert material, to give a mixture that can be processed into coke balls. A highly coking coal can be made into the desired size and quality

of coke balls by this process, without preliminary roasting, by blending with sufficient inert material.

Not only do coals from different fields have different plastic ranges and agglutinating values, but the normal commercial sizing of coal from any mine gives coals with varying plastic ranges. A plant designed to make low-temperature coke by this process from a highly coking coal would be different from a plant designed to treat a weakly coking coal.

Heat Control

Heat transfer has been the fundamental problem in all attempts at carbonization of coal at low temperature. Coal and low-temperature coke are fair insulating materials. In this process the heat for processing is applied both indirectly and directly. Indirect application is through the steel ducts of the roaster and the shell of the carbonizer, and is efficient as long as the metal is clean. There is no difficulty in keeping the metal clean in roaster and storage apparatus, but in the carbonizer, where the atmosphere contains condensable tar vapors, scale accumulates on the steel and retards heat transfer.

It has been pointed out that one factor determining the structure of the coke balls is the rate of heating and maximum temperature in the carbonizing retort. Sufficient heat must be supplied to the coal and breeze mixture in the carbonizer to raise its temperature to about 850°. Higher temperatures than this, for any coal so far examined, only serve to harden the surface of the balls and develop shrinkage cracks. Unless heated to such a temperature (820° to 850°) the product will be too high in volatile matter and will be too soft to handle. Its shatter index will be too low.

To get sufficient heat into the material in process in the carbonizer, "open-end" design and operation have been developed.

Open-end Operation

Using a revolving retort in which coal is fed at one end and coke product withdrawn at the other end, and the gaseous products of distillation are withdrawn at either end, "closed-end" operation endeavors to prevent the entrance of air or other gas. The retort is *closed* and the products are withdrawn; if gas, by suction, and if solid, through sealed outlets. Thus with closed-end operation presumably no reactions take place except those induced by heat carried to the materials in process through the wall of the retort.

The pilot plant at Champion was designed for closed-end operation. The lower end of the carbonizer, however, was provided with an inspection door above the sealed coke outlet, which in operation was kept open for hours at a time. Without intention, the pilot plant was normally

operated "open-end" and no disastrous results followed. This became standard operation when the "star feeder" wore out and was removed.

When the first large retort was designed in 1933 it was provided with an open end, and was so built and operated. However, the significance of open-end versus closed-end operation was not recognized at the time. Unit 3, designed in 1935 and built and operated first in 1936, was designed for closed-end operation, and at the same time closed ends were put on the other retorts. Tight receiving hoppers were installed on the discharge ends, provided with a sliding door, with water seal. Product was collected in this hopper. No air or gas was admitted, except that steam was injected to keep the hot coke from firing while it lay in the hopper. The door was opened at intervals to allow the solid product to drop into the gathering car. There were two serious defects in this system: the gas produced was insufficient to operate the system and the solid product was too high in volatile matter and too soft in structure. There was a gain in production of tar of about 25 per cent and the tar was of better quality. The value of the increase in tar, however, was insufficient to offset the loss in quality of coke and the cost of extra fuel for heating the retorts.

The reason for this difference in operating results, of course, was that it was not possible to put sufficient heat through the shell of the retorts after they became covered with scale. With the lower (discharge) end of the carbonizing retort open to the atmosphere, the by-product gas exhauster is operated so as to pull air into the retort. Slow combustion takes place when the air meets the hot reactive coke and hydrocarbon gases. There is no visible fire. When the change is made from closed-end to open-end operation the by-product gas shows an appreciable increase in carbon monoxide and carbon dioxide and a decrease in oxygen. Because of the air drawn into the open end, the percentage of nitrogen is increased. There is a decrease in yield of tar and a change in the boiling range of the tar. Just where this slow combustion takes place, and what products and in what amounts, are burned, is not definitely known. The evidence indicates that it is the heavy hydrocarbons, the tar vapors, that are oxidized. With the air drawn into the carbonizer in the open-end operation, it has been found necessary also to introduce steam. If steam is not used the coke at the discharge may begin to burn.

That the reactions produced by open-end operation are positive, effective and rapid is shown by a test in Unit No. 2. This unit had been operating closed-end, producing a product with a shatter index of 50 per cent on 2-in. screen. By-product gas production was insufficient to heat the system. The temperature of the coke balls was 800° or less. Within 30 min. after operation was changed over to open-end, the temperature of the balls was raised to 850°, with a shatter index of 70 per cent, and by-product gas was more than sufficient to supply the unit.

Control of the internal heating is by regulation of the "pull" in the by-product gas exhauster.

Operating problems are reduced with open end. When the carbonizer is closed it is necessary to have sensitive automatic controls on the gas mains and exhauster, but with open end no such mechanism is required.

Dams and Lifters

Dams and discharge lifters have been developed for the control of flow and heating of material in process in the revolving retorts. The dams are located and designed to retain beds of material along the length of the retort in such manner as to separate the preplastic and postplastic zones from the plastic zone. In the preplastic zone, in particular, holding material in a bed serves to increase heat transfer to the dry coal. The positive discharge lifter in the nozzle of the retort is a device that permits easy control of the depth of bed at the lower end of the carbonizer. The development of these devices can best be explained by a discussion of operating experience.

If in the smooth sloping steel retort there were no obstructions to flow at any place along its length and no restriction at the outlet, coal fed into the upper end would traverse a length of 100 ft. and be discharged at the lower end in 10 min. In this short interval the coal would be only partly carbonized.

The conical outlet of the retort retains a bed of material the length of which depends on the slope of the retort. In operation, a retort with no dam or obstruction to discharge except the conical outlet builds and maintains a bed of material. The bed is coke balls at the lower and deeper end and coal at the upper end, where it thins out to the minimum. Coal flows down the shell from the charge end in a thin stream until it meets the tail of the bed.

This is the original design of carbonizer. If the design were fortunate as to length of shell, slope, speed, and outlet orifice, the tail of the bed would be in the region where the coal became plastic and balls would be formed. But if the bed were too long and too deep, the balls would destroy themselves by long continual grinding, and would in fact cut the steel of the retort.

A positive lifter was designed for the discharge of product, which by changes in length would afford a means of readily adjusting the maximum bed depth to suit any operating condition desired and would remove the balls as delivered to the lower end of the retort. Changes in the length, or reach, of these positive discharge lifters gives control of the depth of bed of balls and of the length of bed of material in the postplastic stage of the process.

Coal enters the retort at 600° and coke leaves at between 800 and 850°, having passed through a plastic condition in its progress down the retort.

The major heat load in the retort is in heating the coal to its plastic condition. Best utilization of the retort, therefore, calls for maximum heat transfer to the coal in the preplastic stage. But in this stage the coal, and breeze mixed with it, are as a fluid and are conveyed down the retort in a thin, rapidly moving stream. A deeper bed of coal in this stage would have the same effect as to heat transfer from steel to coal as

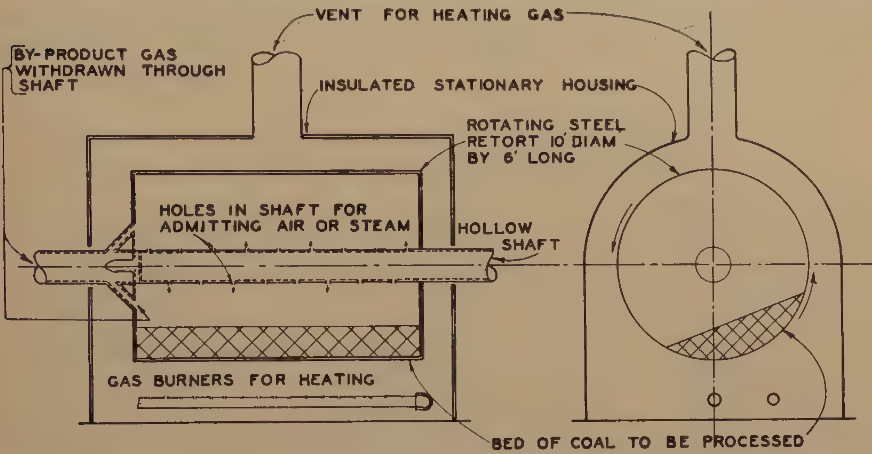


FIG. 5.—APPARATUS FOR CARBONIZING A BATCH CHARGE OF ONE TON OF COAL TO FORM LOW-TEMPERATURE COKE BALLS.

a long retort. The deeper bed of material would be in contact with more area of hot steel, and if stirred and mixed could be heated in shorter length of travel, or more material could be heated in the same length of retort in the same time interval.

Thus another improvement in the carbonizing retort was to install obstructions or dams around the inner circumference of the retort, at a

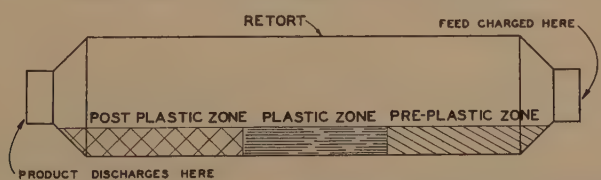


FIG. 6.—DISTRIBUTION OF COAL IN CONTINUOUS PROCESS OF CARBONIZATION IN THEORETICAL HORIZONTAL REVOLVING RETORT.

point down the retort from the feed end where the coal would be heated to a temperature just below its softening point. The three stages, preplastic, plastic and postplastic, are now recognized as distinct, requiring different quantities of heat and of time, and by suitable dams, these three stages are now controlled. Effective heat transfer is made possible by arranging the dams so as to keep coal or material in process on the steel of the retort, more or less over its entire length. As now operated, for instance, the dry, preplastic material is retained in a bed or beds until

it is nearly at the softening point. In the next stage the bed of material becomes plastic and is formed into balls. These balls are not yet hard. The lowest bed is that retained behind the outlet orifice, the depth and length of which are controlled by the positive lifters. Here the balls are hardened and devolatilized to the desired point. The development of this idea is further illustrated by the diagrams in Figs. 5 to 8.

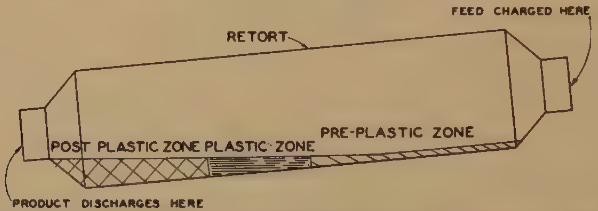


FIG. 7.—DISTRIBUTION OF COAL IN SAME RETORT AS FIG. 5, BUT INCLINED TO MAKE A CONVEYOR OF THE CYLINDER.

The simplest device for making low-temperature coke balls is the batch machine (Fig. 5), described on page 338. In the batch machine each stage of the process is completed before the next begins. Each batch of coal is put through the cycle in one retort. A series of such batch machines, end to end, would make a long retort, and if the operation were so arranged that the material could move progressively through the cycle from dry coal to plastic to coke balls, there would be a continuous operation. Fig. 6 shows such a theoretical retort. It is the horizontal

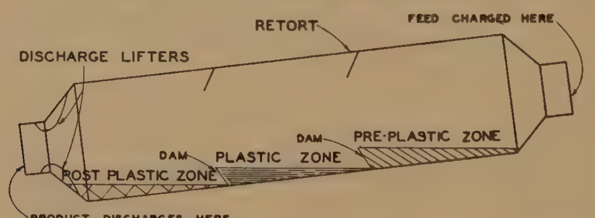


FIG. 8.—EFFECT OF DAMS IN CONTINUOUS REVOLVING RETORT FOR CARBONIZING COAL.

batch machine stretched out to show the complete cycle of operation. The bed of material is of uniform depth from end to end. In practice so long a retort is not operable, because to be continuous the retort must be a conveyor and hence must have a slope, or the material must be higher at the feed end to give a hydraulic head to move it.

In practice the retort is inclined, and the inclination from feed to discharge, with the revolving action, makes of the retort an excellent conveyor. But when so built, the thickness of material, as is indicated in Fig. 7, is much greater at the discharge end. A retort so constructed will be operable for some capacity, but once so erected will be inflexible in its operation. This is the design of Unit 1 as erected in 1933.

As has been described, the practical solution is to build the retort with sufficient slope and to use dams along the inner shell, so placed and

of such height as to separate the successive steps in the process and thus best utilize the heating surface of the retort. Such a design is illustrated in Fig. 8.

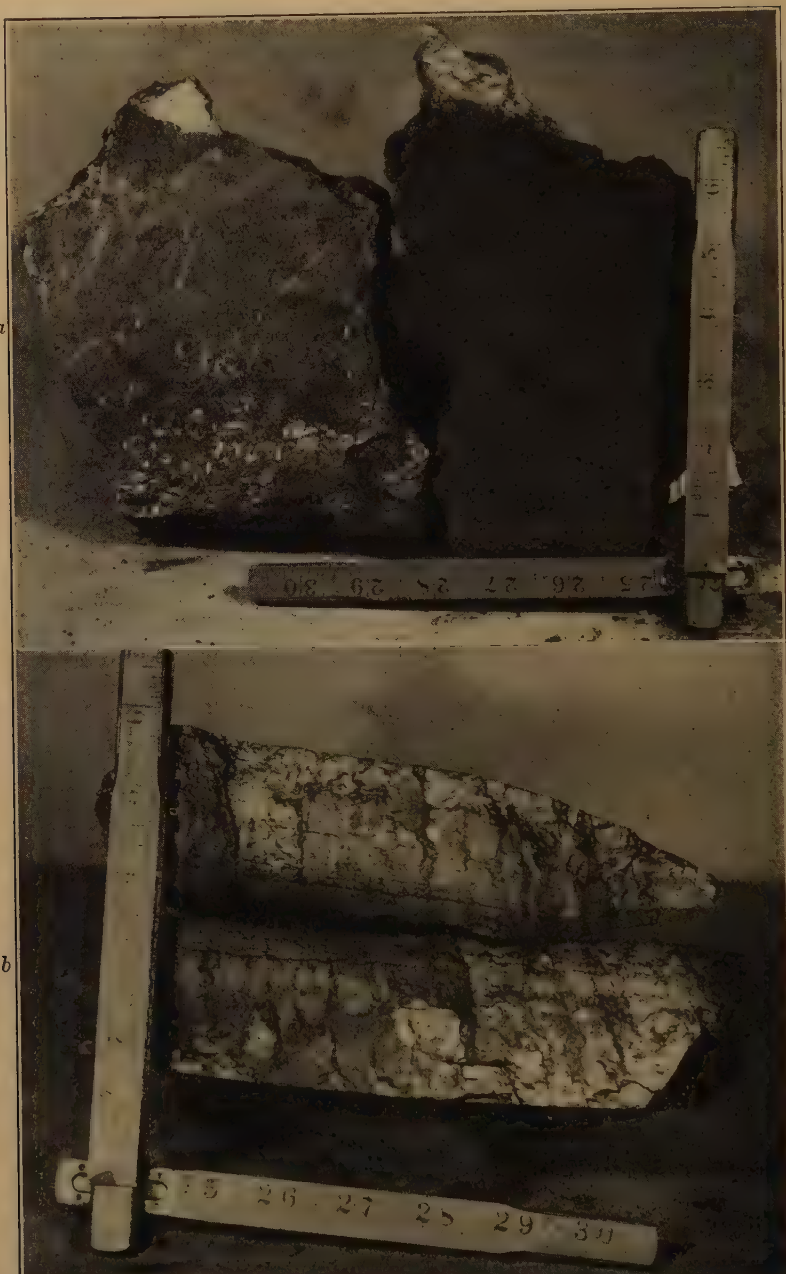
It is very expensive to change the slope of a large retort after it has been erected. However, one with sufficient slope to be an effective conveyor can be adjusted by regulation of dams and lifters so that it will function in the desired manner. The proper adjustment depends upon a knowledge of what takes place in the process and what effect is desired by such dams and lifters.

Scale and Accretions

There is an essential difference in the making of low-temperature coke by this process in the batch machine from manufacture in a continuous plant. This difference is in the accumulation of scale or accretions on the interior of the retort. In the batch machine the reactions are successive in the same shell and each phase of processing takes place in the whole of the retort. The coal and breeze are dry and until the coal softens do not adhere to the steel, but instead polish it. As the coal softens it adheres to the steel, particularly to any irregularities, however small, on the surface of the steel. During the softening phase, when tar vapor is given off in large volume, the mixture becomes plastic and particles of coal in contact with the hot steel are coked and adhere. Once the masses of coke are formed and hardened the rolling of the balls scours the shell and removes all trace of scale. There is no carbon deposit arising from cracking of tar vapor. At the time the tar is evolved, the steel and material in process are at approximately the same temperature.

In the continuous plant coal and breeze are charged at one end of the retort and solid product taken out at the other end. All phases of the process are going on simultaneously and the retort from end to end is filled with an atmosphere of hydrocarbon gas. Heavy tar vapor, from the plastic zone, passing up the retort meets steel at about 800° and comes in contact with coal and breeze at 600° in a very dusty atmosphere. In the lower end, these vapors are in contact with solid coke balls at 800° to 850° and steel at above 950°. In the upper third or preplastic zone of the retort tar vapor is cracked and pitch deposited on the steel in a thin layer, usually not more than ⅛ in. thick. This deposit is as smooth and slick as glass, and as it does not accumulate beyond the initial fraction of an inch, it is not necessary to remove it when the retorts are cleaned.

In the plastic zone a scale is found next to the steel that is the extension of the smooth, shiny scale of the preplastic area. It is thicker, up to ½ in., clings to the steel and is very hard. Overlying it is the "cokey" crust that is characteristic of the plastic zone. This crust is a porous coke that accumulates as the coal is softening and being formed into the masses that make the coke balls. This "cokey" crust may accumulate



a

b

FIG. 9.—SCALE FROM CARBONIZING RETORT. SCALE TAKEN FROM UNIT 3 AFTER 165 DAYS CONTINUOUS OPERATION.

Upper photograph shows flat sides of scale. That on the left was in contact with steel shell and that on the right with the Disco balls.

Lower photograph shows cross sections of same pieces of scale.

to 3 in. in thickness, but after reaching this thickness it breaks off during operation and is nearly all pulverized by the balls in the postplastic zone.

The scale in the lower portion of the plastic zone is very hard and after prolonged operation is between 1 and 2 in. thick. The lower level at which the plastic material hardens is easily identified by the appearance of this hard scale and the absence of "cokey" scale. The hard scale is pitted and rough over the part of the retort in which the balls are soft, but is progressively smoother as the balls become harder and polish it. In the lower part of the retort this scale is never more than ¾ in. thick and is polished and smooth. The hard scale is difficult to remove, particularly in the plastic zone (Fig. 9).

The accumulations in the interior of the retort insulate the shell and decrease heat transmission from the hot combustion gases to the coal in process. Direct heating by the open-end operation offsets this. Progressively as scale builds up inside a retort and heat transmission from the outside is reduced, the open-end heating is increased by drawing more air into the open end.

By-products

The by-products from the low-temperature carbonization of coal vary in kind and quantity, depending upon the type of carbonization process used and the coal carbonized. The by-product yields obtained at Champion will not necessarily hold true for other coals and will vary according to whether the coal requires roasting or a rapid rate of heating to make a satisfactory coke. Roasting and slow rates of heating below the melting point of coal lower the yield of tar and give a somewhat higher yield of coke.

In general, the crude products obtained by the Disco process in addition to coke are tar, light oil, liquor (aqueous liquid) and noncondensable gas. Thus far tar and coke are the only products sold commercially. The gas, containing the light oil vapor, is used in heating the retorts and the excess is wasted to the atmosphere.

Table 5 shows a material balance based on the coal as received. This coal averages about 3 per cent moisture.

The loss includes coal and coke loss in handling, evaporation of liquid constituents, the dust that settles out in the dust collector ahead of the primary tar condenser, and loss in weight of the coal passing through the roaster system. In the roasting system, in addition to loss of surface moisture, there is loss in inherent moisture and water of constitution. During this stage of treatment there is also some loss in the form of carbon monoxide, carbon dioxide and light hydrocarbons.

The rather peculiar fact of the weight per cent of products totaling to more than 100 per cent of the coal is due to the dilution of the byproduct

gas with air, which comes through the seals at the ends of the revolving carbonizers and open-end operation. As is indicated from the analysis of the gas shown in Table 6, approximately two-thirds of the gas comes from air. This accounts for the low B.t.u. value of 414 of the gas, which in a pure state would be as high as 1000 B.t.u.

TABLE 5.—*Yield of Products from* 2000 *Pounds of Coal, as Received*
COMPOSITE OF LABORATORY TESTS AND PLANT PRACTICE

Product	Weight Per Cent	Per Ton Coal
Coke, commercial product	72.0	1440 lb.
Gas, ⅓ from coal, ⅔ from air	12.5	3720 cu. ft.
Tar	7.4	14.75 gal.
Light oil[a]	(0.8)	2.5 gal.
Water: liquor	3.0	7.3 gal.
Vapor to atmosphere	3.4	8.0 gal.
Dust:		
Recovered	3.0	61 lb.
Lost	2.3	46 lb.
Unaccounted for	4.4	
	108.0	

[a] Included in gas.

The illuminants include the various unsaturated hydrocarbons such as ethylene, propylene, butylene and benzene vapor. Methane reported also includes considerable quantities of ethane and traces of propane, butane and pentane. It is estimated that of the 49.9 per cent nitrogen shown in the analysis not more than 0.5 per cent comes from the coal; the rest is the result of air dilution.

TABLE 6.—*Analysis of By-product Gas* (*Unstripped of Light Oil*)

Element	Per Cent	Element	Per Cent
Carbon dioxide (CO_2)	4.2	Hydrogen (H_2)	0.4
Carbon monoxide (CO)	10.0	Nitrogen (N_2)	49.9
Oxygen (O_2)	0.5	Hydrogen sulphide (H_2S)	1.0
Methane (CH_4)	29.5		
Illuminants	4.5		100.0
		B.t.u. per cu. ft., gross	414
		Net	378
		Specific gravity	0.88

It is estimated that the gas can be stripped of its light oils and still supply sufficient heating value to operate the Disco plant. Numerous analyses have shown that the unstripped or raw gas contains light oil equivalent to 2½ gal. per ton of coal feed to the plant.

Light oil from low-temperature carbonization processes differs in constituents and boiling range from that from the high-temperature coking process. The light oil from low-temperature carbonization is comparatively low in aromatics and high in paraffins, naphthenes and olefines. Analyses of the light oil show 10 to 15 per cent of the oil as aromatics (benzol, toluol, etc.), 27 per cent olefines and the balance as paraffins and naphthenes. The oil has a specific gravity at 15.5°C./15.5°C. of 0.760. Its boiling range is unusually low as shown in Table 7. This oil is reported to have exceptionally high antiknock properties and to be valuable as a motor fuel, and may prove to have value as a solvent.

A characteristic peculiar to virtually all low-temperature carbonization processes, including the Disco process, is the absence of naphthalene in both raw gas and tar. A further peculiarity is that the aqueous liquor contains no ammonia in commercial quantities and is acidic rather than alkaline. The liquor is used for direct cooling in a primary condenser and for washing the gas in grid towers for extraction of tar. The acid condition of the liquor is corrected by the addition of lime to prevent corrosion.

Table 7.—*Benzol Flask Distillation of Light Oil*

Temperature, Deg. C.	Per Cent Volume Cumulative	Temperature, Deg. C.	Per Cent Volume Cumulative
27	1st drop	89	50.6
42	5.6	97	61.8
50	11.2	105	73.1
57	16.9	118	84.3
64	22.5	126	89.9
74	33.7	141	95.6
79	39.4	155	98.3
		Dry	100.0

An analysis of one sample of recirculating liquor is as follows: carboxylic acids, 0.6 grams per liter; neutral oil, 5.1; bases, 1.5; phenols, 9.1. This analysis was made by the Coal Research Laboratory of Carnegie Institute of Technology, and is obtained from a private communication from Dr. H. H. Lowry.

The carboxylic acids are believed to be largely acetic acid, although there is probably formic acid, propionic acid, n-butyric acid and n-valeric.

The most important by-product for commercial purposes is the tar. Approximately 1,750,000 gal. of tar are processed annually. In addition to a well equipped laboratory, the tar plant consists of two steam stills with vacuum distillation, a pipe still capable of processing 4800 gal. of tar per day to hard fuel pitch, a fractioning column, and ample storage to take care of blending and seasonal requirements.

The tar produced at the Champion plant is a mixture of primary tar and decomposition products formed in the course of pyrogenic condensation and oxidation in the carbonization process. The pretreatment of the coals in the roasting stage affects the kind of tar produced. Like the other by-products the chemical content and physical properties of the tar vary with the coal being treated. This tar is not orthodox low-temperature tar. Its specific gravity is high, 1.14 at 25°C./25°C. It is rather viscous and not very fluid at room temperature. It has an Engler viscosity of 85 at 50°C. on the dehydrated tar. It is high in insoluble matter, with 18.5 per cent insoluble in CS_2. This is due, not to "free carbon" content, which is under 5 per cent, but to some flocculation of the tar in CS_2 and to dust that travels with the gas stream as it leaves the carbonizer.

TABLE 8.—*Distillation of Crude Tar*[a]

Fraction, Deg. C.	Per Cent by Weight	Cumulative Per Cent
−170	1.3	1.3
235	14.2	15.5
270	9.4	24.9
300	8.5	33.4
300 residue[b]	66.6	100.0
	100.0	

[a] Specific Gravity of the total distillate at 38°C./38°C., 0.984.
[b] Melting point of the plus 300° residue, ring and ball method, 75.4°C.

TABLE 9.—*Distribution of Tar Acids in Tar-oil Distillate*

	Per Cent	
Phenol	7.8	
o-creosol	9.3	
m and p creosols	19.7	
Low-boiling xylenols	16.2	B.p. about 211°C.
Medium-boiling xylenols	5.7	B.p. about 219°C.
High boiling xylenols	5.6	B.p. about 225°C.
Miscellaneous boiling above xylenols	35.7	
	100.0	

A distillation of the crude tar on a dehydrated basis is shown in Table 8. This is a standard A.S.T.M. road-tar distillation made at atmospheric pressure without the use of a reflux or fractioning column.

In the actual operation of the tar stills approximately 50 per cent of the tar comes off as an oil distillate in the making of hard or fuel pitch with a "cube in air" melting point exceeding 175°C. In this distillation liquid temperature in the pipe still reaches 375°C. The oil distillate contains 39 per cent tar acids, of which the greater amount is high boiling.

Table 9 gives the distribution of the tar acids in the boiling range where such materials for commercial use are generally found.

At present there are being made and sold the following tar products: road tars, crude tar acid oil, disinfectant oils, flotation oils, creosote oil (wood preserving), tar paint, and roofing pitch, fuel pitch.

Economics

Undertaken in the first instance as a special problem on a particular coal in a single location, this process has been developed for more than local application. The cost of plant and the cost of operation are within a range that permit conversion of low-value coal into a desirable form of smokeless fuel that has a value in the present highly competitive domestic heating field more than sufficient to pay its way.

The field of this process is not that of high-temperature coke. High-temperature coking is called by-product coking because it depends on the returns—"by-product credits"—from sale of tar, gas, benzol, ammonia, naphthalene and other products. Low-temperature carbonization by the Disco process has no gas for sale, no ammonia to recover, no naphthalene with which to contend. It has tar and light oils, and these are different from those obtained in high-temperature coking.

The field for this process is different from that of high-temperature carbonization, which is primarily to make metallurgical coke and formerly, but to a lesser degree today, to produce gas. This process has been developed to take the lowest priced product of a modern coal mine and make a desirable smokeless household fuel. By this means the coal producer may retain his share of a desirable market that is being rapidly lost to other fuels. A Disco plant differs from a coke plant also in that it may be shut down and started up easily and without damage to the apparatus. It may readily be adapted to fit into a coal-mining and preparation plant and add to the flexibility of such operation.

The physical properties of Disco have been described on page 337. Its bulk density lies between high-temperature coke and bituminous coal. Disco is smokeless, easily ignited, and once ignited does not go out. It holds fire in the ordinary household furnace for several days when the drafts are closed. The housewife likes Disco because it is easily handled, because it is sufficiently reactive to be quickly responsive in the furnace, and because it is easy to build a fire or rescue one that has nearly gone out. In all markets in which it has been sold, Disco has been priced on a competitive basis with high-temperature coke. The distribution of the product is given in Table 10.

Investment in plant depends on the kind of coal used, and the size and location of the installation. The kind of coal influences cost because strongly coking coal requires equipment for pretreatment that is not required for weakly coking coal. Coal, coke, gas, and tar-handling

equipment are common to all plants and all coals and are here designated "outside" plant.

Estimates of capital cost for plants of one, two, three and four units are given in Table 11. These are costs for plants using strongly coking coal that must be pretreated. The estimates are for complete, self-contained installations, and are based on costs of construction of the present plant. In Table 12 are estimates for plants of like capacity but designed for coking coal that does not require roasting preliminary to carbonizing.

TABLE 10.—*Geographical Distribution of Disco Sold*

State	Tons		
	1937	1938	1939
Michigan	1,575	2,172	2,901
Ontario	7,609	8,791	5,649
Maryland, Virginia, and District of Columbia	473	464	516
New York and New England	3,438	3,723	3,327
Indiana	193	233	301
Ohio	2,995	4,417	5,292
Pennsylvania	19,517	33,688	45,253
	35,802	53,488	63,241

TABLE 11.—*Investment Cost of Disco Plant Using a Strongly Coking Coal That Requires Roasting*

LIGHT OIL RECOVERY AND TAR DISTILLATION NOT INCLUDED

	One Unit	Two Units	Three Units	Four Units
Carbonizer, roaster and storage	$154,000	$297,000	$434,000	$566,000
Outside plant	$110,000	$143,000	$160,000	$176,000
Total plant	$274,000	$440,000	$594,000	$742,000
Capacity, tons of coal per day	160	320	480	640
Tons of coal per 300-day year	48,000	96,000	144,000	192,000
Depreciation, cents per ton of coal[a]	36.7	30.6	27.5	26
Plant cost per ton daily coal capacity	$1650	$1375	$1235	$1160

[a] Depreciation figured on 15-year life of plant. Total dollars plant cost, divided by 15 and by tons coal carbonized per year.

The influence on cost of size of plant and kind of coal is shown in these figures. Outside plant costs for a one unit plant are $690 per ton daily capacity and $276 for a four-unit plant. Processing equipment, including roasting for a four-unit plant, is estimated to cost $880 per ton of coal per day capacity and $550 for a coal that does not require roasting.

Depreciation is figured on coal capacity per year of 300 days and 15 years as life of plant.

The estimates in Tables 11 and 12 include railroad tracks, roads, truck scales, wash house, office, warehouse and accessories necessary for continuous operation. A different situation arises when a unit is to be operated in conjunction with an operating mine or coal-preparation plant, where certain facilities are already at hand and only carbonizing equipment and coke and tar-handling machinery are required. In such a situation the throughput is limited to the mine output of the particular size of coal to be used. The operating personnel of the Disco plant is restricted to that necessary for the actual production and making of Disco. It is not necessary here to provide skilled mechanics for maintenance work because the limited requirements can be furnished from the mine organization. In similar manner the clerical work, the payrolls and shipping of product can be carried by the established mine organization.

TABLE 12.—*Investment Cost of Disco Plant Using Coal That Does Not Require Roasting*

LIGHT OIL RECOVERY AND TAR DISTILLATION NOT INCLUDED

	One Unit	Two Units	Three Units	Four Units
Carbonizing retort	$100,000	$187,000	$270,000	$352,000
Outside plant	$110,000	$143,000	$160,000	$176,000
Total plant	$210,000	$330,000	$430,000	$528,000
Capacity per day, tons of coal	160	320	480	640
Capacity per year of 300 days, tons of coal	48,000	96,000	144,000	192,000
Depreciation, cents per ton coal carbonized	29.2	22.9	19.9	18.3
Plant cost per ton daily coal capacity	$1312	$1030	$895	$825

The cost of carbonizing retort and Disco cooling and handling plant necessary for 100 to 150 tons of coal per day, when attached to existing mine or preparation plant, will not exceed $1000 to $1200 per ton-day capacity, as compared with $1300 to $1650 for a small self-contained plant.

For efficient and low-cost operation, this type of plant should be operated continuously at least 300 days per year. To meet the requirements of an isolated mine operation where the objective is processing of some certain portion of the output of small size of coal, and where it is desirable to process the fine coal currently, an intermittent retort has been designed. Such a retort can be operated currently with the mine, working full time or part time as desired. Such a plant will have substantially lower cost of installation but higher operating cost than the continuous plant. An intermittent retort with a minimum of accessories will cost, it is estimated, $90,000. For a coal that requires no roasting, this retort will have a capacity of 5 tons of coal per hour, or 120 tons per 24 hr.

Operated 300 days per year, depreciation would be 17¢ per ton of coal charged; operated 200 days, 25¢.

Conversion Cost

Labor, supplies and power are the items that make up cost of conversion. Of these labor cost is the largest and the one that varies the most. Labor costs as given here are based on wage rates from 64 to 78¢ per hour and the weighted average cost of a man for 8 hr. is $6. The foreman or superintendent is figured at $10 per day. Labor for the self-contained plant (costs given in Tables 11 and 12) includes a superintendent, three shift foremen, clerk and chemist or sampler. For example, such a two-unit plant would have six men per shift in addition to the salaried force. Labor cost is estimated at 41¢ per ton of coal. A four-unit plant would have a labor cost of 30¢ per ton of coal. A two-unit plant operated in conjunction with a mine organization would have a labor cost of 25¢ per ton of coal.

A single retort of the intermittent type has a higher labor cost, between 70 and 75¢ per ton, but with two such units, the cost is estimated at between 40 and 45¢ per ton, if operated with the producing mine.

Supply cost is more directly a function of throughput, and is between 10 and 12¢ per ton of coal, exclusive of cost of steam. A plant processing 500 tons of coal per day requires about 75 boiler horsepower of low-pressure steam.

Power cost depends on unit cost of energy. The continuous plant has a high load factor. The load factor at Champion is 88 per cent and the power factor 90 per cent. Power cost can be estimated from the requirement of a maximum of 25 kw-hr. per ton of coal feed, at from 10 to 16¢ per ton.

Conversion costs, then, range from 80¢ per ton of coal for a continuous self-contained plant with capacity of 640 tons of coal per 24 hr. to $1.36 for a similar plant with capacity of 160 tons. Such a plant, with capacity of 160 tons, operated in conjunction with a mine plant will have an estimated conversion cost of $0.93; and with capacity of 320 tons, $0.75 per ton of coal. The intermittent plant with capacity of 120 tons per day, operated at a mine, will have an estimated conversion cost of $1.24 and with 240 tons per day capacity, a cost of $0.95.

Yields

It has been stated (Table 5) that the yield of Disco at Champion is 72 per cent of the coal charged. Over the four years 1936 through 1939 the actual salable Disco over 1-in. screen was 71 per cent of the coal charged to the plant. Many coals have been tested here with commercial yields that range from 60 per cent for high-oxygen, high-moisture coals to 73 per cent or more for higher rank coals that do not require roasting.

The yield of Disco must be considered in translating the conversion costs into a profit and loss operating statement. Also there must be considered the yield and value of tar. Tar yield at Champion is 14.75 gal. per ton of coal charged. This is a minimum for coals tested, largely because of the destruction of hydrocarbons in roasting. Other high-rank coals from which Disco is made without roasting give 20 or more gallons per ton of coal. The low-rank, high-oxygen, high-moisture coals yield 14 to 17 gal. per ton of coal.

Value of Products

Disco sells f.o.b. cars at the point of production at prices ranging from two to three times the cost of the coal required for its production. For instance, if a ton of coal to be used in making Disco costs $1.25, and yields 70 per cent Disco, the coal to make a ton of Disco costs $1.79. The Disco sells for $4.50, or 2.5 times the cost of the coal used. A certain coal, valued at 50¢ a ton, yielding 60 per cent Disco, or 83¢ for the coal to make a ton of coke, will sell in that location for $3.00, more than three times the cost of the coal. Another coal, valued at $1.00, will yield 70 per cent coke, which has a value of $3.50 per ton at the ovens. The ratio here is 2.45. Generally, the coals that are most suitable for making Disco are the fines that in recent years have been marketed at less than a dollar a ton. The market value of Disco is now $4.50 f.o.b. ovens, in the East, and will meet competition anywhere in the eastern half of this country at an average over the year at from $3.25 to $4.00. The margin between coal cost and Disco sales realization exceeds $2.00 per ton of product.

The value of the tar from producing a ton of Disco is from 60¢ to a dollar; adding to the $2.00 an average of 75¢ for tar, gives $2.75 to pay for conversion, overhead and profit. Thus in a general way is indicated the economic warrant for production of Disco.

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DISCUSSION

(*H. E. Nold presiding*)

H. H. Lowry,* Pittsburgh, Pa.—This description of the Disco plant is interesting and supplies the answers to many questions that have been asked. However, there are a few points on which additional information seems desirable.

Fig. 3 shows the rates of cooling of coke balls, ranging in size from 4 to 8 in. in diameter. Since the initial temperatures are so nearly independent of size of ball, it would be interesting to learn whether the temperatures given are surface temperatures or center temperatures. If they are the latter, it must indicate that essentially all the heat is transferred to the coal before the balls are actually formed and that after the formation of the balls devolatilization takes place in the retort at approximately constant temperature.

From data published by the Bureau of Mines on assay tests of Pittsburgh-seam coals at 500°C., the yields of coke reported appear low and are not entirely consistent in the paper. The yield of commercial coke of 72 per cent presumably includes all the breeze that is fed back into the charge to the carbonizer. Yet the data on ash content of the coal used and the Disco obtained in Table 3 would indicate a coke yield of 78.4 per cent.

The discussion on agglutinating index of coals as a measure of their suitability for use in the Disco process suggests two questions: What range in values of agglutinating index may coals have and still be satisfactory raw materials for the process? Is the function of the roasting operation to reduce the agglutinating index to a certain value before the charge is fed to the carbonizer?

In high-temperature carbonization liquor disposal is a real problem, the liquor frequently being used to quench the coke. Since in the Disco process quenching is not practical, what methods of liquor disposal are used?

The yield of light oil given in Table 5 is much higher than reported by the Bureau of Mines for Pittsburgh-seam coals carbonized at 500° and suggests that the methods of analysis used may be different. Was the figure reported the light oil scrubbed from the gas with wash oil?

H. C. Porter,† Philadelphia, Pa.—The coal industry, carbonizing industry, and all engineers and scientists interested in study of coal-carbonizing problems are greatly indebted to Mr. Lesher and his sponsoring company for this paper, which gives the practical fundamentals underlying a successful low-temperature carbonizing operation. The Disco is one of but two or three low-temperature operations that have made any headway in this country, and it alone has gone successfully to a commercially important scale of operation. No doubt this achievement is due to the great amount of thorough and careful development work devoted to it by the author and his associates.

As intimated in the early part of the paper, there should be a large field for the application of this process to the upgrading into salable high-grade domestic fuel of the large amounts of fine screenings now obtained in various mining districts as an incidental product of bituminous preparation by the modern methods.

On the economic side, although the margin of profit is small (as indicated by the author's figures in the latter part of the paper), it appears that the operation described, on Pittsburgh coal, provides a satisfactory return on the investment, and any plant

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with similar yields and on a similar scale can do so if operated steadily on at least an 83 per cent load factor (300 days per year). This allows $1.25 per ton for the coal, and affords the mines a steady outlet for their otherwise difficultly salable screenings.

In estimating costs and returns, it should be noted that the paper (Table 11) gives depreciation and investment costs on a per-ton-coal basis, as also, on next following page, the conversion costs are given, while the sales values of coke and tar, given on page 357, are on a per-ton-coke basis, and confusion would be likely to arise therefrom in making comparisons. The sales values of coke and tar cannot, of course, be put on a per-ton-coal basis in general for all coals, since percentage yields vary widely, but assuming Pittsburgh coal, of the Champion grade, the author's figures appear to show, on a two-unit plant carbonizing 320 tons coal per day, 300 days per year, the following:

Returns, per ton coal treated (assuming 70 per cent yield of coke):		
Coke, at average of $3.63 through year	$2.54	
Tar, 14.7 gal. at 0.035	0.52	
		$3.08
Costs:		
Coal	1.25	
Conversion, varying with conditions	1.05	
Depreciation	0.31	
		2.61
Balance, per ton coal treated		$0.47

This balance is the net return on an investment of $4.58 per ton coal, or slightly over 10 per cent, which is to be reduced further by taxes, insurance and miscellaneous overhead. As the author states, these figures vary with different coals and the kind of operation needed. The margin, apparently, is small, and would be reduced still more with any material reduction of load factor or operating time per year, not only through increase per ton in investment costs, but also in conversion and depreciation per ton. Larger-scale operation, on the other hand, would increase per-ton profits and yield on the investment. Even with a small margin of profit, the important, highly advantageous feature that this process offers is its provision of a steady outlet for the fine coal screenings at a favorable price.

On the technical side, it is evident from the paper, and in fact, well known, that the product, Disco, is an unusually strong and dense low-temperature coke, with a high factor of combustibility and ease of ignition. Although the paper states (page 341) that structure is in part due to "the character and proportion of inerts," it remains somewhat obscure whether good structure and strength require the admixture of inerts (fine coke breeze) with the coal entering the carbonizer up to the amount used; that is, 22 per cent of the mixture. It would be of interest to know whether any trial runs have determined whether, without addition of breeze, or with a reduced quantity, Disco would still have its good strength and density and size of pieces, or whether they would be materially affected. The author states (page 337) that in current operations at the Champion plant there is a deficiency of breeze for good operation of the carbonizers, amounting to about 2 per cent, but it is not clear whether such a deficiency of breeze affects seriously the quality of product.

In high-temperature carbonization, in coke ovens, the admixture of 3 to 5 per cent of fine coke breeze in the charge, especially with coals of a high degree of plasticity

and high swelling index (under free expansion), is beneficial to the coke size and strength. Since the Pittsburgh coal seam, in this locality, has been shown by the Bureau of Mines (*Monograph* 5) to have a higher than average free-expansion or swelling index—namely, 1.7 to 2.4—it is to be expected that, when carbonized, at low temperatures, without confining walls and without addition of inerts, it will tend to puff up into light, porous char or semicoke. Preliminary roasting, as Mr. Lesher says, reduces this tendency by altering the nature of the highly plastic material, but whether this would be enough to give the density and strength of Disco with use of only a small percentage of added inerts remains unanswered. At any rate, the upgrading in this way of all of the low-value breeze and dust is a highly advantageous feature.

As to the ease of ignition of Disco, compared to that of coal (see page 337), the method used in determining this quality will influence the comparison. Unquestionably Disco is a highly reactive material and has a low ignition point, but when comparing with coal or wood or other material that is easily altered by heat and oxidation during the test procedure, a method that avoids or at least minimizes this error in determining ease of ignition gives a truer comparison than methods that permit alteration of the sample before it reaches the ignition point. This reviewer, using a method designed to minimize such error, has compared Pittsburgh-seam coal with a fused char made from it at 750° to 770°F., and found the latter slightly more reactive, but found commercial low-temperature cokes (two samples, not Disco) less reactive than the coal from which they were made. A report on these results and this method has been made to the American Chemical Society, Fuels Division, but not yet published.

Another point as to the ignition quality of Disco is the chemistructural one concerned with the absorption of oxygen by the coal material during the preliminary roasting. When the material is gently carbonized, following the roasting, only a part of this oxygen probably is driven out again, and what remains, very possibly, contributes to the ease of ignition of the residue coke. This reviewer, in studies of relative ignitibility of solids (mentioned above), has shown that combined oxygen, of certain structural types, appears to increase generally the factor of ease of ignition.

An inquiry suggests itself in connection with the by-product tar from Disco, with reference to the high phenolic content, especially of the higher phenols—whether disposal of such tars or tar fractions is possible for the manufacture of phenolic resins.

It is suggested that in figuring the quantity of heat used in this process for conversion of the coal (pp. 334–335) there must be added to the heat input (in heat balance, Table 2), given as 538 B.t.u. per pound of feed, the unknown heat produced in the roasting process as well as any exothermic heat developed in the carbonizer. The total might then be close to 800 B.t.u. per pound of coal.

C. E. Lesher (author's reply).—Dr. Lowry and Dr. Porter have raised some interesting questions that should be answered to make the record clear. Information in the paper on coke yield at the Champion plant is unfortunately confusing and should be explained in further detail. The coke yield cannot be accurately determined by calculation from the ash and volatile content of the raw coal and the Disco because of the loss of high-ash fines in the roaster and in the washbox dust. The laboratory shows a theoretical recovery on the coal being used at Champion of 76 per cent of coke with 18 to 19 per cent volatile; whereas operations over a period of years show 71 per cent of merchantable product shipped. Table 5, showing the yields of products, has been revised from the original paper to better present the facts as known. The percentage of light oil shown in Table 5 was recovered from plant gas with dry ice. Use of wash oil would have given lower yield of light oil from gas. The liquor is

circulated, cooled, and then used again for cooling gases and in that way has a large loss by evaporation. What is not evaporated is wasted.

Present knowledge indicates that coals with agglutinating index between 2.5 and 5 or 6 will make Disco without pretreatment. Roasting preceding carbonization is to reduce the agglutinating property if too high.

The record given of cooling rate on Disco balls is based on center temperatures. Dr. Lowry is correct in saying that most of the heat in the product is in the ball when it is formed. Dr. Porter raises the question with respect to the usefulness and value of inerts in the mixtures being fed to the carbonizer. Good coke structure requires the admixture of inerts, in this case pulverized breeze, regardless of whether the coal requires roasting ahead of carbonizing or not. For a coal that requires roasting, the control of the product depends upon the amount of roasting and the proportion of inerts. For a coal that does not require roasting, control depends upon the rate of heating and the proportion of inerts. The size range of the inerts is important. If the pieces are too large or if they are extremely fine, the structure of the Disco is not satisfactory.

There is still much to be learned about the heat balance of the process and the data given in the paper are the best that are now available.

M. D. Curran,* St. Louis, Mo.—To those who have read through this paper it is evident that a great amount of painstaking effort was required to present the data in such an informative manner. I have been very much interested in the development Mr. Lesher has undertaken and can appreciate as well perhaps as anyone the courage and determination it has taken on Mr. Lesher's part to bring this process to its present state of completion. Any one familiar with the action of coal at so-called "low temperatures" will understand something of the problems involved, which he had to overcome. His success in solving these problems is evidenced by the fact that the sale of Disco in markets tributary to his plant has made possible enlargement of his plant facilities to their present capacity.

We have approached the problem of domestic fuel manufacture from a somewhat different angle, which employs standard so-called "high-temperature" treatment of the coal in such a way as to provide the carbonized product with a much higher reactivity than has been obtained heretofore through the carbonization of coal in vertical by-product coke ovens. Our coal charge is spread in a relatively thin layer about a foot thick on a horizontal floor and heated from beneath at standard high temperatures, but in this method of heat-treatment a different coke substance is produced, which in effect has burning characteristics similar to so-called "medium temperature" coke. We generally get about 50 per cent more tar than is produced in other high-temperature processes and a correspondingly smaller amount of gas. The treatment of coal in this fashion has permitted the production of a fairly strong coke substance from coals that normally are classified as of the feebly coking variety. In the production of domestic smokeless fuel, operations are generally carried on in such a fashion as to produce a fuel with 2.5 or 3 per cent volatile matter, whereas where metallurgical coke is required the operations are adjusted to maintain the volatile matter between 1 and 2 per cent.

My experience convinces me that a broad demand is on the way for smokeless fuels. This, we think, is due to the determination of people living in large cities to eliminate the damaging effects of smoke produced through the use of bituminous coal in hand-fired house-heating equipment. The coal industry is conscious of this demand through the increasing requirements of coal especially prepared for the domestic

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stoker. The use of household stokers measured by sales reported by manufacturers of such equipment shows that people are beginning to realize the advantage of automatic heat produced through the use of coal. Programs under way in many of our large cities for smoke elimination have been creating a strong demand for stoker-sized coal, which may be expected to increase as time goes on. Smoke will be eliminated by all people that own their own homes and can afford to buy mechanical equipment for burning bituminous coal smokelessly, but in every large city many people rent their homes and smokeless fuel suited to their present heating equipment must be furnished if smoke is to be completely eliminated.

Submission of data concerning developments is important to the mining industry because it is apparent that coal producers in certain sections of this country are going to be interested in beneficiation of the small-sized screenings resulting from the production of domestic stoker coal, so that this part of their production can be marketed as a domestic smokeless fuel.

H. L. Griffin, Morgantown, W. Va.—Mr. Lesher's paper is interesting from a number of different angles. It demonstrates what can be done by those who intelligently go ahead with the object of solving a problem of making a waste product valuable. The increasing demand for smaller stoker and pea sizes leaves the resultant fines to be disposed of. Giving these sizes value through low-temperature carbonization tends to raise the coal operators' realization, which should put him in a definitely better competitive position in the energy market.

C. A. Reed,* Washington, D. C.—The fact that this fuel contains enough volatile may have something to do with the continuous banking ability. There is also some form within the coke cells that is difficult or impossible to describe, which, perhaps partly because of its high porosity, makes within the coke itself a sustaining oxygen supply. I say this is a theory because there does not seem to be any other more plausible answer and, in fact, the coke does act that way when the furnace is tightly closed, when a day's supply of ash is allowed to accumulate on the grate and when apparently no outside air is supplied to the burning fuel. The fact that this fuel can be left in a furnace for an extremely long period with a tight bank and that it will continue to burn until there is practically no fuel left, therefore only the ash residue, indicates that there is some peculiar action within the coke cells. The fact that this product does burn under continuous bank and must have oxygen to burn appears to indicate that there is oxygen or a synthetic gas within the coke structure itself. I realize that this is a rather poor explanation and perhaps further study on the burning of this product may give a more practical answer. I understand that that answer is not available today, however.

C. C. Russell,† Pittsburgh, Pa.—It is gratifying to find a low-temperature process in successful operation after so many failures. The reactivity of low-temperature coke is most often cited as one of its principal advantages in use, but this same advantage may also be considered as a disadvantage. Because of its high reactivity, low-temperature coke will produce a considerable amount of carbon monoxide in the flue gases, particularly where deep fuel beds are involved. Ignition of such flue gas may cause explosions that will blow open the fire door and ash-pit door. The question is raised as to how much difficulty has been encountered in the sale of Disco from this source and what firing directions are supplied to the householder so that he may avoid such occurrences.

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C. E. Lesher (author's reply).—Virtually no difficulty has been encountered in the sale of Disco because of explosions that will blow open the fire door and ash-pit door of the furnace. There is much less difficulty in this respect with Disco than with bituminous coal, either high-volatile or low-volatile, and hardly any more than with high-temperature coke or anthracite. The simplest rule for all firing in hand-fired household furnaces is to leave a hot spot showing while firing, and whenever that is done there is never any difficulty from explosions. And when that is *not* done there are few such explosions ever reported to any of our dealer customers or transmitted to us.

Proportions of Free Fusible Material in Coal Ash, as an Index of Clinker and Slag Formation

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(New York Meeting, February 1940)

The softening temperature of coal ash, as determined in the laboratory, has been used for years as an indication of the tendency of coal to form clinker and slag. It has not, however, provided an index of unfailing and complete reliability for forecasting the relative performance of different coals, even in the same plant and under the same conditions of operation. A large amount of investigation has been done in attempts to account for this, but practically all of this has been in the direction of studying the chemical constitution of coal ash. Much has been learned, but without much progress toward a better practical application of knowledge to the selection of coal. There has always seemed to be an important factor that eluded discovery.

It has long been recognized that the ash-softening temperature as determined in the laboratory had one weakness, in that it represented the temperature at which a thorough *mixture* of the whole ash of the coal would melt, the mixing having taken place before the heat was applied. This does not, of course, simulate what takes place in practice. With certain exceptions, which will be noted later, almost all of the investigations that have been made, especially the chemical studies of coal ash, have likewise dealt with a mixture of the whole ash in the coal.

The present investigation differs from these fundamentally in its approach to the problem. It started from the observation that some clinkering and slagging obviously takes place in a boiler furnace at temperatures below the laboratory-determined ash-softening temperature of the coal. Knowing that coal as it is shipped to market is a heterogeneous mixture of diverse mineral substances, each of which may act independently of the others in the initial stages of clinker and slag formation, it seemed logical to investigate separately the softening temperature of the ash in physically separable portions of the coal, and to determine the relative quantities of ash contributed by each portion.

While the data are far from complete, covering as they do only a few coals, and while the conclusions to be reached from the data at this stage are, in some measure, speculative, they are presented at this time because

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they offer sufficient promise of fruitful results to encourage further study along the same lines by others.

Two methods of investigation were used, one aimed to throw light on the formation of clinker in a solid fuel bed; the other relating to slag formation in a pulverized-coal furnace. The identification of the coals studied and the analysis of the head samples are given in Table 1.

TABLE 1.—*Identification of Coals and Analyses of Head Samples*

Coal	Seam	County	State	Analysis (Dry Basis), Per Cent					
				Volatile	Ash	Sulphur	British Thermal Units	Initial Deformation Temperature, Deg. F.	Ash-softening Temperature, Deg. F.
A.......	Lower Freeport	Jefferson	Pa.	28.0	9.9	0.8	13,760	2800+	2800+
B.......	Pittsburgh	Westmoreland	Pa.	29.7	8.4	0.9	14,295	2780	2800+
C.......	Upper and Lower Freeport	Indiana	Pa.	31.2	6.3	0.7	14,535	2600	2700
D.......	Upper Freeport	Armstrong	Pa.	31.8	9.3	1.2	13,635	2480	2580
E.......	Lower Freeport	Clearfield	Pa.	29.9	8.5	1.2	14,140	2400	2480
F.......	Thick Freeport	Allegheny	Pa.	35.9	6.5	1.1	14,370	2360	2460
G.......	Pittsburgh	Monongalia	W. Va.	35.8	7.3	2.1	13,965	2080	2140

METHOD OF INVESTIGATION IN RELATION TO CLINKER FORMATION

In the study related to clinker formation, each coal was reduced to approximately the same top size, from 95 to 100 per cent through a 4-mesh screen. Either 1¼ or 2-in. nut-slack coal was used. The crushing to 4-mesh was adopted in the first place to conform to the special requirements of an investigation of slagging in a particular case, and was later adopted as a suitable size for an extension of the study with relation to underfeed firing, on the assumption that this size would roughly approximate the size of the coal when it reached the fuel bed. Just to what extent the coal is crushed in the stoker varies with the friability of the coal and the type of stoker, but there is no question that considerable crushing action takes place.

In order to secure a maximum of differentiation of materials, each sample, after crushing, was separated into four gravity fractions, and each of these into six or seven size groups, as follows:

GRAVITY	SIZE, MESH
Float 1.3	+4 (if any)
1.3–1.4	4–8
1.4–1.5	8–16
Sink 1.5	16–30
	30–50
	50–100
	−100

TABLE 2.—*Proportion of Coal and Ash and Initial Deformation Temperature for Each Fraction, in Coal Crushed to 4-mesh*

Gravity	Coal / Head sample	A			B			C			D			E			F		
	Ash, per cent	9.9			8.4			6.3			9.3			8.5			6.5		
	Initial deformation temperature	2800°+			2780°			2600°			2480°			2400°			2360°		
	Ash-softening temperature	2800°+			2800°+			2700°			2580°			2480°			2460°		
	Size, Mesh	Coal, Per Cent	Total Ash, Per Cent	I.D.T., Deg. F.	Coal, Per Cent	Total Ash, Per Cent	I.D.T., Deg. F.	Coal, Per Cent	Total Ash, Per Cent	I.D.T., Deg. F.	Coal, Per Cent	Total Ash, Per Cent	I.D.T., Deg. F.	Coal, Per Cent	Total Ash, Per Cent	I.D.T., Deg. F.	Coal, Per Cent	Total Ash, Per Cent	I.D.T., Deg. F.
1.3 float	+4	1.4	0.5	2800+	1.1	0.5	2800+	1.1	0.6	2800+	1.3	0.5	2800+	0.5	0.3	2800+	0.8	0.5	2780
	+8	9.2	3.2		8.3	3.9		9.8	4.8		5.7	2.1	2800+	9.0	5.0		16.8	9.1	
	+16	11.1	3.4	2800+	12.9	5.5	2800+	14.9	6.3	2800+	9.6	2.5	2800+	18.9	9.2	2800+	15.7	7.1	2740
	+30	11.8	3.0	2800+	12.9	4.9	2800+	14.9	5.1	2740	11.0	2.3	2700	20.4	9.1	2760	14.5	5.5	2700
	+50	9.1	2.2	2800+	8.3	2.6	2800+	9.9	2.9	2680	7.1	1.3	2700	12.1	4.7	2740	10.1	3.2	2660
	+100	6.7	1.4	2660	5.1	1.5	2800+	6.3	1.8	2640	5.0	0.9	2580	7.8	2.9	2700	7.0	1.9	2580
	−100	5.7	1.2	2480	3.9	1.2	2800+	5.4	2.5	2380	3.9	1.0	2560	9.3	4.7	2300	6.6	2.2	2400
	Total	55.0	14.9		52.5	20.1		62.3	24.0		43.6	10.6		78.0	35.9		71.5	29.5	
1.3–1.4	+4	1.3	1.1	2800+	1.8	2.3	2800+	0.9	1.3	2800+	3.3	3.3	2800+	0.1	0.2	2800+	0.6	1.0	2800+
	+8	6.8	5.5		9.6	12.0		6.9	9.5		8.9	9.0	2800+	2.0	3.0		6.2	8.9	
	+16	5.8	4.6	2800+	10.0	12.3	2800+	7.6	10.2	2800+	9.4	9.2	2800+	3.4	4.9	2800+	4.5	6.6	2800+
	+30	4.6	3.7	2800+	7.4	9.0	2800+	5.7	7.4	2800+	7.6	7.2	2800+	3.1	4.3	2800+	3.2	4.8	2800+
	+50	3.0	2.3	2800+	4.0	4.7	2800+	3.0	3.9	2800+	4.3	4.0	2800+	1.7	2.3	2780	1.9	2.7	2800+
	+100	2.2	1.7	2800+	2.4	2.7	2800+	1.6	2.1	2800+	2.8	2.5	2800+	0.9	1.3	2700	1.1	1.6	2800+
	−100	4.0	2.6	2560	2.6	2.5	2800+	2.6	2.4	2620	4.5	2.8	2600	1.0	1.0	2680	1.8	1.8	2580
	Total	27.7	21.5		37.8	45.5		28.3	36.8		40.8	38.0		12.2	17.0		19.3	27.4	
1.4–1.5	+4	0.7	1.3	2800+	0.4	1.0	2800+	0.3	0.8	2800+	0.3	0.7	2800+	0.1	0.1	2800+	0.1	0.4	2680
	+8	2.0	3.9	2800+	1.8	4.4		1.3	4.0		0.8	1.8	2800+	0.4	0.9		1.2	3.6	
	+16	1.1	2.0	2800+	1.6	3.8	2800+	1.3	3.7	2800+	0.9	1.9	2760	0.5	1.2	2800+	0.8	2.6	2740
	+30	0.8	1.4	2800+	1.1	2.7	2800+	1.0	2.7	2800+	0.9	1.8	2800+	0.4	1.0	2760	0.7	2.0	2780
	+50	0.5	0.8	2800+	0.6	1.5	2800+	0.6	1.6	2800+	0.7	1.3	2800+	0.2	0.5	2600	0.4	1.2	2740
	+100	0.4	0.6	2800+	0.4	0.9	2800+	0.4	1.0	2800+	0.6	1.0	2800+	0.2	0.4	2580	0.3	0.7	2800+
	−100	0.7	0.8	2600	0.4	0.6	2800+	0.6	1.1	2660	2.1	2.2	2660	0.2	0.4	2200	0.4	0.7	2700
	Total	6.2	10.8		6.3	14.9		5.5	14.9		6.3	10.7		2.0	4.5		3.9	11.2	
1.5 sink	+4	1.8	9.5	2760	0.2	1.2	2440	0.1	1.1	2140	0.5	2.5	2400	0.3	1.9	2500	0.1	0.6	2020
	+8	3.4	17.2	2720	0.8	4.9	2260	0.8	4.9	2120	1.0	5.2	2100	1.6	9.1	2240	1.0	5.6	2000
	+16	1.8	8.5	2680	0.7	4.0	2140	0.8	5.1	2120	1.3	6.7	2100	1.8	9.8	2180	1.0	5.7	1980
	+30	1.2	5.6	2660	0.6	3.1	2160	0.7	4.4	2100	1.4	6.8	2080	1.4	7.7	2140	0.9	5.6	1980
	+50	0.9	3.8	2400	0.4	2.2	2160	0.5	3.1	2160	1.1	5.1	2080	0.9	5.0	2160	0.7	4.5	2020
	+100	0.7	3.2	2280	0.3	1.8	2120	0.4	2.4	2160	1.0	4.6	2080	0.8	4.0	2060	0.6	4.1	2100
	−100	1.3	5.0	2200	0.4	2.3	2080	0.6	3.3	2100	3.0	9.8	2060	1.0	5.1	2020	1.0	5.8	2040
	Total	11.1	52.8		3.4	19.5		3.9	24.3		9.3	40.7		7.8	42.6		5.3	31.9	
	Grand total	100.0	100.0		100.0	100.0		100.0	100.0		100.0	100.0		100.0	100.0		100.0	100.0	

This resulted in either 24 or 28 portions of each sample separated according to the size and density of the particles. Each portion was then ground to 60-mesh, and tested for percentage of ash and the ash-softening temperature. By this procedure it becomes possible to determine the *fusibility* of the ash by classes of material, which are capable of acting independently in the fuel bed, and also the *proportion* of the whole ash in the coal, which is contributed by each portion. It also throws light upon the *distribution* of fusible material. These are all factors in clinker formation, which do not seem to have received heretofore as much attention as they deserve. These factors are important because, if the easily fusible material in the ash is already mixed with the refractory material within individual pieces of coal, the practical effect in the fuel bed can be expected to differ from that which would take place if the same relative amount of fusible material were concentrated in separate particles of fuel, which might become segregated, and whether they did or not, would be capable of independent action in *starting* the formation of a clinker.

Fusibility of Ash in Physically Separable Portions of Crushed Nut-slack

The data for the six coals studied given in Table 2, show that coals having substantially the same ash-softening temperatures determined in the conventional way have strikingly different proportions of fusible material, and significant differences in its distribution.

This is more easily seen, when the data are summarized as in Table 3,

Table 3.—*Proportions of Total Ash in Each Range of* 100° *Initial Deformation Temperature for Coal Crushed to 4-mesh*

Coal	Head Sample A.S.T., Deg. F.	Under 2100°	2100°–2199°	2200°–2299°	2300°–2399°	2400°–2499°	2500°–2599°	2600°–2699°	2700°–2799°	2800° and Over
A....	2800+			8.2		5.0	2.6	16.3	26.7	41.2
B....	2800+	2.3	11.1	4.9		1.2				80.5
C....	2700		24.3		2.5			8.2		65.0
D....	2580	26.3	11.9			2.5	1.9	5.0	5.5	46.9
E....	2480	9.1	22.5	9.5	4.7		2.3	1.5	21.3	29.1
F....	2460	27.8	4.1			2.2	3.7	6.8	28.2	27.2

which shows the proportions of the whole ash within each 100° range of temperature of initial deformation. We have used throughout the Initial Deformation Temperature (I.D.T.) instead of the Ash-softening Temperature for the individual fractions, because this study seems to emphasize the importance of the independent action of the ash in separable portions of the coal in *starting* clinker formation. A study of these data suggests certain conclusions:

1. These six coals would be arranged in nearly the same order, according to their expected clinker-forming tendencies, on the basis either of the ash-softening temperature of the whole ash or according to the amount and distribution of the fusible material in physically separable portions of the coal, thus supporting the common experience that the accepted laboratory determination of ash-softening temperature is a very useful, but not infallible, index of this coal characteristic.

2. The exceptions, however, are even more interesting, and probably more important, in suggesting why performance does not always agree with predictions based upon the conventional determination of ash-softening temperature, especially for two coals having the same ash-softening temperature.

Table 4.—*Proportions of Total Ash, Classed as Fusible, Refractory and Intermediate, for Coal Crushed to 4-mesh*

Coal	Head Sample A.S.T., Deg. F.	Fusible (Under 2300° I.D.T.), Per Cent	Intermediate (2300° to 2600° I.D.T.), Per Cent	Refractory (Over 2600° I.D.T.), Per Cent
A.......	2800+	8.2	7.6	84.2
B.......	2800+	18.3	1.2	80.5
C.......	2700	24.3	2.5	73.2
D.......	2580	38.2	4.4	57.4
E.......	2480	41.1	7.0	51.9
F.......	2460	31.9	5.9	62.2

For convenience in discussion, this may be more readily seen, if the material in Table 3 is further condensed as in Table 4, in which the ash is broadly classified as fusible (under 2300°), refractory (over 2600°), and intermediate (between 2300° and 2600°). While both coal A and coal B would be rated at 2800°+ by the standard laboratory method, coal B contains more than twice as much fusible ash, so distributed as to be capable of independent action in starting a clinker. Coal C, while rated at 2700°, has 33 per cent more fusible material in a free state than coal B.

The difference of 100° in the conventional fusing-point rating between coals D and E suggests a greater difference in clinker-forming tendencies than would be arrived at from the relatively small difference in the proportions of free fusible materials. The more detailed data in Table 3 show that coal D has a much larger proportion of the ash having an I.D.T. above 2800°, which probably accounts for the higher *average* ash-softening temperature of the whole ash, but this may have no practical effect on clinker formation, when the proportion of fusible material, which would start the clinker formation in the first place, is very nearly the same in both coals.

The difference disclosed between coals D and F is more striking. Coal D, with an ash-softening temperature 120° higher, contains a substantially higher proportion of free fusible material. Coals E and F, while having substantially the same ash-softening temperature (2480° and 2460°), differ substantially in the amount of free fusible material, coal E having 41.1 per cent of the whole ash having an I.D.T. below 2300°, and coal F having only 31.9 per cent of its ash in that classification. These two coals might, therefore, be expected to behave quite differently under identical fuel-bed conditions.

Another general conclusion suggested by these data is that, in many coals at least, the ash in different portions of the coal, when the coal has been crushed to this size, is sharply divided into two distinct classes, that which is readily fusible (usually below 2200°) and that which is highly refractory (usually above 2700°). This further suggests that the small proportion of the ash that has an intermediate I.D.T. represents an accidental mixture of the two chief components, which has happened to be held together in some individual particles of coal.

TABLE 5.—*Weighted Average Percentage of Ash in Coal in Each Gravity Fraction*

CRUSHED TO 4-MESH

Gravity	A	B	C	D	E	F
1.3 float	2.8	3.3	2.4	2.3	3.9	2.9
1.3–1.4	8.0	10.4	8.2	8.9	11.6	9.3
1.4–1.5	17.9	20.4	17.1	16.0	19.3	19.0
1.5 sink	48.9	48.8	39.6	41.7	46.9	39.5
Weighted average	10.2	8.6	6.3	9.5	8.4	6.5

PULVERIZED

Gravity	A	B	C	E	F	G
1.3 float	2.0	2.7	2.1	2.3	2.3	2.7
1.3–1.4	5.3	6.0	4.6	6.2	4.2	5.7
1.4–1.5	15.5	17.5	11.7	10.5	16.0	12.3
1.5–1.6	19.5	25.6	21.5	16.5	22.0	16.1
1.6–1.7	28.4	32.3	27.1	25.1	25.8	23.5
1.7–1.8	35.2	40.5	33.3	28.8	38.2	28.2
1.8–1.9	42.9	46.9	44.6	40.5	43.9	43.4
sink 1.9	70.8	67.9	61.8	63.8	62.5	64.4
Weighted average	9.8	8.0	6.3	8.2	6.3	7.3

It will be seen from an examination of Tables 3 and 4, that even in coals having a high ash-softening temperature substantial quantities

of free fusible material for the initial formation of clinkers may be present, and that in coals like D and F, having softening temperatures of 2580° and 2460° respectively, more than 25 per cent of the total ash is easily fusible (I.D.T. below 2100°), and is present in physically separable portions of the coal.

It would appear, therefore, that in burning this type of coal on a stoker, the most significant factor in the early stages of clinker formation may be the relative quantity of free fusible material. All of the ash having an I.D.T. below 2300°, in all of these coals, was found in the 1.5 sink, and this material contained from 40 to 50 per cent ash (see Table 5). Thus the free fusible material occurs in a highly concentrated form, capable of becoming nuclei of numerous individual clinkers, with which the more refractory portions of the ash ultimately become combined, in varying proportions. Under such circumstances, segregation prior to and during firing may easily play a part in determining the size and structure of the final clinker formation.

It would seem that this or some similar method of measuring the relative quantity of free fusible material offers more promise of providing a more reliable indication of relative clinkering characteristics than the softening temperature of the whole ash, especially for discriminating among coals having an ash-softening temperature of 2400° and higher.

In order to arrive at the proportion of free fusible material, it is necessary to make both the size and gravity separations. This can be seen clearly in connection with coal A (Table 2). If a mixture of all the ash found in the 1.5 sink material had been tested for softening temperature, the presence of the more refractory ash in the larger sizes would have obscured the presence of the much more fusible ash in the smaller sizes. The same thing is true of size classification alone.

The chemical composition of the two classes of ash would probably help to explain the composite effect reflected in the ash-softening temperature as found in the laboratory, and of course can throw some light on the type of clinker or slag resulting from the final mixture of these materials, but it does not and cannot reveal the relative amount of free fusible material capable of *starting* clinker formation, which this fractional study of ash attempts to determine.

Fusibility of Ash in Physically Separable Portions of Pulverized Coal

For this portion of the investigation, samples of the coal were pulverized to pass a 200-mesh screen and then separated into a number of gravity fractions from 1.3 to 1.9. The gravity separation of 200-mesh coal was carried to 1.9, because it was found that it was still possible to separate small quantities of refractory material between 1.5 and 1.9. The fact that by crushing coal the impurities are freed is a matter of

TABLE 6.—*Proportions of Coal and Ash, and Initial Deformation Temperature for Each Gravity Fraction, 200-mesh Coal*

	A			B			C			E			F		
	Coal, Per Cent	Whole Ash, Per Cent	I.D.T., Deg. F.	Coal, Per Cent	Whole Ash, Per Cent	I.D.T., Deg. F.	Coal, Per Cent	Whole Ash, Per Cent	I.D.T., Deg. F.	Coal, Per Cent	Whole Ash, Per Cent	I.D.T., Deg. F.	Coal, Per Cent	Whole Ash, Per Cent	I.D.T., Deg. F.
Float 1.3......	4.9	1.0	2560	24.7	8.3	2800+	16.3	5.4	2440	35.0	9.9	2640	34.3	12.6	2600
1.4..........	73.7	39.8	2800+	62.0	46.8	2800+	72.3	53.2	2800+	49.8	37.6	2800+	56.1	37.3	2780
1.5..........	12.1	19.2	2800+	7.6	16.7	2800+	6.3	11.9	2800+	5.5	7.0	2800+	3.7	9.5	2800+
1.6..........	3.8	7.6	2800+	2.5	8.1	2800+	2.0	6.9	2800+	3.9	7.9	2800+	1.6	5.6	2800+
1.7..........	0.8	2.5	2800+	1.0	4.0	2800+	0.9	3.7	2800+	0.9	2.8	2800+	0.8	3.3	2800+
1.8..........	0.6	2.0	2800+	0.5	2.3	2800+	0.4	2.4	2800+	0.7	2.4	2800+	0.5	3.1	2800+
1.9..........	0.6	2.7	2800+	0.2	1.2	2800+	0.3	2.0	2700	0.2	1.0	2800+	0.4	2.5	2800+
Sink 1.9.......	3.5	25.2	2200	1.5	12.6	2060	1.5	14.5	2260	4.0	31.4	2040	2.6	26.1	2120
	100.0	100.0		100.0	100.0		100.0	100.0		100.0	100.0		100.0	100.0	

common knowledge, and one that is applied in the preparation of coal. One result of this study is to show that there is progressive separation of impurities at each successive stage of crushing, all the way down to sizes as small as 200-mesh.

This can be seen by an examination of the data in Table 6, which gives the I.D.T. for the ash in each gravity separation of minus 200-mesh coal for five of the six coals, which also were studied in the minus 4-mesh size. With the exception of the ash in the 1.3 float, the ash in all of the other fractions down to the 1.9 sink is definitely of a highly refractory nature, so that there is a much sharper division between the fusible and the refractory material than was found in the minus 4-mesh coal. This would indicate that, in the main, the ash in these coals is composed of two distinct classes of materials, one highly refractory and the other easily fusible, and that these two components of the ash are not only physically separable but are separated in the process of grinding. This confirms the conclusion that the fractions separated in the minus 4-mesh size that contained ash having intermediate softening temperatures were combinations of these two materials in varying proportions.

From the standpoint of pulverized-fuel firing of these coals, from 12.6 to 31.4 per cent of the total ash in the coal is fusible material introduced into the furnace in separate particles in a highly concentrated form, consisting of 60 to 70 per cent of incombustible material. The remainder of the ash, except a small portion found in the 1.3 float in some of the coals, is highly refractory.

The fusible material, being in separate particles, quickly melts in the furnace and adheres to walls and tubes, forming a sticky film. A decreasing quantity of the fusible material is present in the gases as the distance from the burner increases. The refractory ash adheres to the film of fusible material in varying proportions, according to the distance from the burner, thus producing a variety of slags having different physical properties. Slag removed from superheater tubes is often found to be composed almost entirely of refractory material bound by a very small proportion of fusible ash.

The conclusion, therefore, would seem to be that the most important factors in predetermining the slagging characteristics of a coal of this general type in a pulverized-coal furnace is the *amount of free fusible material*, and the relative amounts of fusible and refractory materials.

One coal having an ash-softening temperature of 2140° was studied in pulverized form (coal G, Table 8). This is obviously an entirely different type of coal from the standpoint of either clinker or slag formation. If any refractory material is present, it is not physically separable, and apparently all of the ash is easily fusible. This type of coal might be expected to produce a more nearly uniform type of slag at all points in the furnace.

TABLE 7.—*Proportion of Pulverized Coal and Percentage of Ash in Coal in Each Gravity Fraction*

Gravity	A		B		C		E		F		G	
	Coal, Per Cent	Ash in Coal, Per Cent	Coal, Per Cent	Ash in Coal, Per Cent	Coal, Per Cent	Ash in Coal, Per Cent	Coal, Per Cent	Ash in Coal, Per Cent	Coal, Per Cent	Ash in Coal, Per Cent	Coal, Per Cent	Ash in Coal, Per Cent
Float 1.3...	4.9	2.0	24.7	2.7	16.3	2.1	35.0	2.3	34.3	2.3	24.4	2.7
1.3–1.4....	73.7	5.3	62.0	6.0	72.3	4.6	49.8	6.2	56.1	4.2	65.2	5.7
1.4–1.5....	12.1	15.5	7.6	17.5	6.3	11.7	5.5	10.5	3.7	16.0	4.3	12.3
1.5–1.6....	3.8	19.5	2.5	25.6	2.0	21.5	3.9	16.5	1.6	22.0	2.2	16.1
1.6–1.7....	0.8	28.4	1.0	32.3	0.9	27.1	0.9	25.1	0.8	25.8	0.5	23.5
1.7–1.8....	0.6	35.2	0.5	40.5	0.4	33.3	0.7	28.8	0.5	38.2	0.6	28.2
1.8–1.9....	0.6	42.9	0.2	46.9	0.3	44.6	0.2	40.5	0.4	43.9	0.2	43.4
Sink 1.9...	3.5	70.8	1.5	67.9	1.5	61.8	4.0	63.8	2.6	62.5	2.6	64.4
	100.0		100.0		100.0		100.0		100.0		100.0	
Weighted average..		9.8		8.0		6.3		8.2		6.3		7.3

It is also interesting, as shown in Table 7, that in firing coals of the kind included in this investigation in pulverized form, from 80 to 90 per cent of the coal is introduced into the furnace in the form of particles that contain less than 6 per cent ash all of which, except in coal G, is of a refractory nature, while only 1.5 to 4.0 per cent of the coal consists of particles that contain from 60 to 70 per cent of ash of which all is fusible at low temperature.

TABLE 8.—*Proportions of Coal and Ash and Initial Deformation Temperature for Each Gravity Fraction of Coal G, Pulverized to* 200-*mesh*

Gravity	Coal, Per Cent	Whole Ash, Per Cent	I.D.T., Deg. F.
Float 1.3..................	24.4	9.2	2360
1.4..................	65.2	50.7	2300
1.5..................	4.3	7.4	2120
1.6..................	2.2	4.9	2060
1.7..................	0.5	1.5	2160
1.8..................	0.6	2.2	2100
1.9..................	0.2	1.4	2060
Sink 1.9..................	2.6	22.7	[a]
	100.0	100.0	

[a] Not determined.

RELATION OF RESULTS OF TWO METHODS OF INVESTIGATION

Comparison of the data obtained from the minus 4-mesh study and that obtained from the separation of the minus 200-mesh material reveals

certain points worth noting. Table 9 shows the percentage of total ash classed as free fusible material (below 2300° I.D.T.), as determined for

TABLE 9.—*Comparison of Percentage of Free Fusible Material (Below 2300° Initial Deformation Temperature) in Ash in Crushed and Pulverized Coal*

Coal	Crushed to 4-mesh, Per Cent	Pulverized to 200-mesh, Per Cent
A	8.2	25.7
B	18.3	12.6
C	24.3	14.5
E	41.1	31.4
F	31.9	26.1

the two sizes. Coals B, C, E and F have a smaller proportion of free fusible material in the minus 200-mesh size, which seems to indicate that some refractory ash remained combined with the fusible ash in the larger size, and this was released by grinding. Coal A, on the other hand, has a very much larger proportion of fusible material in the pulverized coal, indicating that the fusible material was so widely and evenly distributed through the coal structure that it had little effect upon the softening temperature of the ash of the various fractions in the larger coal. Only by fine pulverization was the fusible material finally released. This seems to be borne out by the fact that coal A is definitely differentiated from the others by the high softening temperature of the ash in the larger sizes of the 1.5 sink (Table 2).

OTHER INVESTIGATIONS

There have been numerous investigations of the relation of ash-softening temperature to clinker and slag formation. Several of these approached this subject from a direction somewhat similar to the present one, but with certain differences in method, which should be noted. Nicholls and Selvig[1] in their extensive study, separated 20-mesh coal only at 1.35 gravity. The free fusible material, for a number of the coals at least, was still mixed in the sink with refractory ash contained in the coal particles, which would float somewhere between 1.35 and 1.9, with the result that there was not the same sharp separation between fusible and refractory ash.

Moody and Langan[2] studied the components of the ash, by making size and gravity separations of the ash itself, which indicated somewhat similar differences among coals, but this method does not relate the constituents of the ash to the coal particles that carry them.

McCabe and Rees[3] made size and gravity separations of Illinois coals, but without crushing or pulverizing them. Except for this, their approach to the problem was very similar to the one described here.

[1] References are at the end of the paper.

The coals they were studying, however, all had a very low ash-softening temperature (in the neighborhood of 2000°), so that their results would not reveal the same wide range of fusibility in the several fractions.

Bailey[4] approached the subject through a study of the fusibility of the slag collected at different points in the furnace, rather than from the standpoint of the separable constituents of the ash contained in different classes of coal particles. He recognized, however, that individual ash particles, having widely different softening temperatures, would act independently in causing slag in a pulverized-coal furnace. His investigation is the complement of this one, in that he was studying the effect, while we were studying the cause, in an attempt to find a satisfactory way to predetermine the effect.

Assuming that a determination of the proportion of free fusible material will furnish a more reliable index of clinker-forming characteristics among coals having also a substantial proportion of free refractory ash than the standard method of determining the ash-softening temperature of the whole ash (a point yet to be established by experience and observation in the field), the best method to be followed is debatable. Whether the sample should be crushed to pass a 4-mesh screen, or some other size, or whether it should be separated without crushing, is an important question, which needs serious consideration. It seems to us, however, that the separation should be made in approximately the size of the coal at the time it reaches the fuel bed. The method described here is offered as a starting point.

For the determination of slag-forming characteristics in pulverized-coal firing, the size of the coal at time of firing is known and can be closely approximated before the separation takes place. As stated in the beginning, the results of the present investigation are presented, not as a complete or conclusive study, but one that is suggestive of a new and promising method of approach to the problem, which we hope will be further investigated by others.

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DISCUSSION

(L. I. Cothern presiding)

E. G. Bailey,* New York, N. Y.—This paper brings out vividly the segregation of the diversified constituents of ash in coal. The authors properly present this as a progress report, without finalizing their recommendations as to its usefulness in tying in the results of the laboratory with the actual use of the coal.

The authors refer to a paper that I published in 1938, in which I gave certain data relating to the fusing temperature range of the slag and ash collected from separate parts of a boiler furnace, these relating particularly to units fired with pulverized coal. It happens that the data shown in my paper, Figs. 5 and 7, related to coal similar to that listed by Gould and Brunjes as coal G. Table 10 gives some comparative data from the coal ash, and from the segregated ash by the float method by Gould, and similar data from my paper and from subsequent tests from the same coal and furnace, the latter showing segregation of the ash within the boiler furnace. Of the 26.5 per cent iron in the tapped slag, 19 per cent of it was Fe_2O_3, while of the 14 per cent iron in the slag from the boiler tubes, 79 per cent of it was Fe_2O_3.

The question of segregation is extremely important in connection with pulverized coal. The ash collected on walls of the hotter zone of the furnace is almost invariably higher in iron content than is the ash in the coal itself, while the iron content of the sponge ash that may accumulate in the superheater, and perhaps the boiler-tube banks, is generally about the same as that in the ash of the coal, while the fly ash, which passes on through, is lower in iron.

Another phase of this ash segregation, which has been brought out in studies that we have carried on most extensively and diligently in the past year subsequent to the paper, is that in a boiler furnace and the boiler-tube bank the form of the ash may be quite different from that resulting from laboratory determination by the A.S.T.M. method. I refer particularly to the form of the iron, as to whether it is fully oxidized to Fe_2O_3 or reduced to FeO.

In boiler furnaces, we find that the ash tapped from the furnace as molten slag is highly reduced—that is, a large percentage of FeO—while the ash accumulating on the upper furnace walls and tube banks, especially in the superheater, may be highly oxidized so that most of the iron is in the form of Fe_2O_3. There is a difference in many coals of as much as 300° to 500° between the initial deformation on these two bases.

There is not only segregation of iron in the slag in furnaces, but there is a varying degree of oxidation of the iron in the slag, so that these two factors together take us very far afield in actual practice from the laboratory determination from a sample of coal. It is believed that the next step is to make further extensive studies in the laboratory, checking the conditions in furnaces where the latter are tied back to the rates of heat absorption and the effect upon the operation of the boiler in a very fundamental way. At the moment I am rather skeptical as to whether or not we can redeem fusing temperature from laboratory samples of coal as a useful method of indicating, with any degree of accuracy, certain important features of these coals with respect to their operation in different types of furnace. This is said not in a critical attitude, but only to spur on everyone who is working on this subject to carefully coordinate their activities with the opinions of others, so that the entire subject may be thoroughly explored, and that should be done promptly.

I urge that we continue with the A.S.T.M. method, reporting all three temperatures instead of curtailing the report to the Initial Deformation or Softening Tempera-

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ture. It is recommended that the iron content of the ash or slag be determined, and preferably the fusing-temperature range be run in an oxidizing atmosphere also.

TABLE 10.—*Comparative Data*

	Gould and Brunjes	Bailey	
Coal	G, Table 1	Fig. 7	Latest Test
Initial deformation, deg. F.	2080	2020	2070
Softening temperature, deg. F.	2140	2105	2140
Fluid temperature, deg. F.		2420	2510
Fe in ash, per cent			14

	Float	I.D.T., Table 8	I.D.T.	S.T.	Fluid Temperature	Atmosphere in Laboratory, Fusing-temperature Determination	Iron in Ash as Fe, Per Cent	Slag from Furnace at
Lowest	1.9	2060	2020	2100	2600	Reducing	26.5	Slag tap
Highest	1.3	2360	2390	2470	2580	Oxidizing	26.5	Slag tap
			2150	2200	2510	Reducing	14	Boiler tubes
			2350	2410	2590	Oxidizing	14	Boiler tubes

P. NICHOLLS,* Pittsburgh, Pa.—One would expect that data on coal-ash distribution for quantity and fusibility, such as those presented by the authors, should permit more accurate prediction than values for the average ash only, but the application of such data will not be simple. The data themselves are complex, and although the authors have analyzed them in several ways still further comparisons are possible. For example, in Table 4, for the crushed 4-mesh coal, the fusible material for coal B is greater than that for coal A, 18.3 against 8.2 per cent; however, the reverse is true on the basis of Table 6 for the 200-mesh coal, in which the fusible material of A is 25.2 per cent against 12.6 per cent for B; thus there is question of the justification for predicting that B would clinker more than A.

It will be interesting to see how such data can be usefully applied to fuel beds. Distinctions because of the distribution of the fusible material will be greater as the rate of burning and the temperature of the bed are low, because there will be less possibility of the high-fusion combining with the low-fusion ash. As the rate is higher, and as the size of the fuel is smaller, these more exact data on the ash will be less important because more of the high-fusion and low-fusion material will combine, and the cone values for the average ash would be truer measures.

We have shown (ref. 1 and Bur. Mines *Bull.* 378) that in underfeed burning the ash is well mixed and is subjected to high temperatures for a longer time than in overfeed burning; consequently, the actions to which the ash is subjected more nearly resemble those of the cone test. Thus for clinkering in underfeed stokers one would expect that the fusibilities of subdivisions of the ash would not be of much greater value than that for the average ash.

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In our work on clinkering (ref. 2) we did not find any better correlation between the measures of clinkering used and the fusibilities of the float and sink of 1.35° gravity than with the softening temperature of the average ash.

It would be expected that these data would be most useful for pulverized fuel because the particles have a chance of acting independently of one another. Thus there can be selective actions in that the particles that fuse have a better chance of being deposited on the slag bed or surfaces, and the lighter particles carried through the furnace. A study of the amount of this selective action was the original object of the work we did in cooperation with power stations.[4] Table 2 of the reports shows the amount of the selective action for each station; its average value was low but was large in some instances.

The authors ask how the coal should be sized if such tests were standardized. Studies for the distribution of quantity and fusibility of ash in the various sizes of coal pieces made by McCabe and Rees[3] gave useful information, but for use in the burning of coals it would suffice to use a pulverized size. This sizing would be required for pulverized burning, and there is no reason why a larger size would be more correct for fuel beds. In the burning of coal there is no assurance that all the ash of one piece of coal remains together, or that there is distinction between the ash of adjacent pieces. The combinations of ashes that occur in a bed are largely a matter of chance in the arrangement of the pieces.

The separation of coal into density ranges by float-and-sink of the pulverized size is the extreme of such classification, but it does not follow that all the pieces in a narrow range of density will have the same fusibility. If all the organic matter and all the ash each had a constant density, the float-and-sink process would separate the coal into ranges in each of which the ash content of all particles would be the same but their fusibility might vary over the full range. Actually the ash varies in density and, because iron is the usual flux, fusibility decreases with increase in density. This fact tends to segregate the ash by fusibility in the float-and-sink separations, but it is still possible that the fusibility of the particles in a narrow range of density may vary widely.

The authors use the initial deformation instead of the softening temperature. The justification for this is questionable, but it would not affect their comparisons because the interval between the two temperatures is about the same for all their coals; in other coals the interval varies widely. That such a large proportion of the ashes tested had an initial temperature of plus 2800°F. is remarkable; with the A.S.T.M. gas-furnace method ashes of that initial value do not contain more than 4.5 per cent Fe_2O_3 and 2 per cent CaO.

It is very desirable that studies of this type should be continued, but in these initial stages they should include the chemical analyses of the ashes. The thoroughness of the work being done by the Illinois State Geological Survey is commendable, and they promise the chemical analyses corresponding to the data they have already published. It would be helpful if, in addition, the mineralogical forms of the ash in the coals could be determined, because they might help to elucidate some otherwise unexplainable values.

There should be caution against expecting too much from data of this type. Our recent paper[5] on the viscosity of coal-ash slags shows that the properties of a fused ash are not necessarily defined by its three cone-fusion temperatures, and certainly not by the initial alone.

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[4] P. Nicholls and W. T. Reid: Slags from Slag-tap Furnaces and Their Properties. *Trans.* Amer. Soc. Mech. Engrs. (1934) **56**, 447–465.

[5] P. Nicholls and W. T. Reid: Viscosity of Coal-ash Slags. Preprint, Amer. Soc. Mech. Engrs. (October 1939).

H. D. Bowker,* Omar, W. Va.—It is a known fact that the softening temperature of coal ash is an unreliable index for forecasting the performance of different coals in relation to formation of slag or clinkers, even when burned in the same plant under identical conditions. Much attention and experimenting have been spent in studying the chemical constituents of coal ash, even to the extent of being able to approximate the softening temperature of a coal ash, based on the effect of the different chemical constituents. As has been stated, there is a noticeable contrast in dealing with mixed samples of ash and a coal that is referred to as a heterogeneous mixture of mineral substances.

An experience several years ago proved to us that the softening temperatures of coal ash as determined in a laboratory were false in indicating what could be expected of one of our coals in a firebox relative to clinker or slag formation. On a slack coal the softening temperature of ash as determined by the laboratory showed 2400°F. However, in actual performance clinkers and slag formations were reported at lower temperatures. Samples of slag and clinker taken disclosed a black, glossy, tarry appearance.

Investigating the consist of the slack, we found that the trouble lay in the minus 20 mesh; the 20 by 48-mesh as well as the minus 48 mesh having an ash-softening temperature of 2160°F. The ash in the minus 20-mesh represented 16 per cent of the ash of the slack.

An examination of the minus 20-mesh disclosed amounts of fusain ranging from 40 to 65 per cent, the larger amounts of fusain being found in the finer mesh sizes, being highest in the minus 200-mesh fraction. While the removal of the minus 20-mesh dust only increased the softening temperature of the ash approximately 100°F., in actual performance the coal withstood temperatures over 300°F. higher than previously it did without clinker or slag formation. It is also an interesting fact that we have not experienced the same kind or type of clinkers.

Several years have passed since we began removing the minus 20-mesh dust and it can be truthfully stated that complaints due to this particular coal clinkering, as before, have been nil.

R. A. Sherman,† Columbus, Ohio.—Inasmuch as the writer has on numerous occasions, both privately and publicly, maintained that the standard method for the determination of the fusion characteristics of coal ash is in error because it attempts, by pulverization and through mixing, to make homogeneous a material that is heterogeneous both in the coal and the fuel bed, it is needless to say that he commends the work that the authors present in this paper. The data that they present confirm the heterogeneous nature of coal ash for coals A to F and the exception, coal G, serves to prove the rule. Coals A to F were remarkably alike in that the greater part of the ash was quite refractory with initial deformation temperatures of 2700° to 2800°F. and most of the more fusible constituents had initial deformation temperatures of 2100° or lower. Very little of intermediate fusion temperature was found. Perhaps this is a general rule but it cannot be predicted definitely and further similar studies of more coals will be required to prove or disprove it.

The writer would not accept without question the authors' assumption that the initial deformation temperature is a better index than the ash-softening temperature, particularly in view of the general experience that the latter point is more easily read with accuracy. He also would question the size to which the coal was crushed for the study. The 4-mesh size may have been in order for the coals studied but, in general, for stoker application the top size would be larger for firm-structure, high-

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volatile coals. It should be, as the authors say, the size of the coal that reaches the fuel bed.

For pulverized coal, the studies should be made on coal as pulverized for burning rather than all through 200 mesh, and separations should be made on the coal by sieving and elutriation as well as by density. Such a separation on some coals would be expected to show readily fusible material in the coarse sizes because of the presence of pyrites.

A correlation of the action of the ash in a stoker fuel bed or pulverized-coal-fired furnace with a study such as this, showing the type of clinker, the wall slagging, tube fouling, and the carry over of ash into the passes and stack would be much more convincing of the value of this type of study than the laboratory results alone. It is to be hoped, however, that the results shown here, which add to the previous work of Moody and Langen, McCabe and Rees, and Bailey, will lead to such correlated studies.

This type of investigation is not simple. It requires many separations and many determinations. But the problem of clinkering and slagging of coal ash is not a simple one and it is not to be expected that the solution can be by simple methods.

R. G. Pfahler,* Windber, Pa.—The deviations in quality that occur in various shipments of coal from any mine can be expected to extend to the various increments that were segregated by screening and gravimetric methods for the basic data presented in this paper by Gould and Brunjes. As Mr. Gould has shown in numerous papers, a series of samples, systematically taken, is necessary to develop the true characteristics of a given size coal from any mine. While the variations in quality shown by individual samples of commercial coal are often the result of the samples containing varying percentages of the different size and gravimetric increments, it is believed that investigators that have carefully studied the washability characteristics of coals will agree that several such studies of the same coal will show incremental deviations somewhat similar to those that occur in commercial analyses. This indicates that the method suggested by Gould and Brunjes requires considerably more analytical work to carry the study to its ultimate conclusion.

For obvious reasons, the authors have relied on the initial deformation temperature of the ash to compare the relative ash-fusion characteristics of the different increments derived from each sample. In view of the fact that all three critical points —the initial deformation temperature, the ash-softening temperature and the fluid temperature—are frequently necessary to fully define ash fusibility, these critical temperatures should be reported when they can be determined in the testing furnace. Fusion tests on a wide variety of coals will show some with a low I.D.T. having a relatively high A.S.T. This may be true of some of the increments segregated by Gould and Brunjes.

It is evident that the nature of the coals chosen for the tests reported is such that the pure coal has a high I.D.T., while the coal dust, in some cases, and the impurities (1.5 sink), in every case, have a relatively low I.D.T. In some coal seams, the nature of the mineral matter and pure coal is such that the fusion characteristics as shown in this paper will be reversed; that is, the pure coal ash will fuse more readily than the impurities.

It is doubtful whether segregation in a furnace will be any more perfect than that which occurs in pneumatic coal-cleaning processes where classification is far from perfect. Those several increments that might then be closely associated in the initial stages of clinker and slag formation should have their ash-fusion characteristics determined by actual test in composites of different proportions. In extending the study along these lines, particular attention should be given to possible eutectic effects.

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* Berwind-White Coal Mining Co.

The suggested approach to the problem of clinkering suggested by the paper warrants further study. However, other factors than the ash-fusion characteristics of coal influence clinkering and slag formation in a furnace; it would seem, then, that with the detailed study of different coals combustion tests to correlate the proportions of free fusible material with the clinker and slag would be essential to determine whether performance could be predicted.

R. D. Hall,* New York, N. Y.—This admirable paper shows that, at least in general, fine and dirty parts of the coal have an ash of the lowest deformation temperature, and the paper by D. R. Mitchell [Segregation in the Handling of Coal. *Trans.* A.I.M.E. (1938) **130**, 107–128] shows that the fine and dirty parts of the coal cohabit the same section of the coal bin. Holmes [*Colliery Engineering*, 1934] showed the same with regard to separation when coal is dumped to form a cone.

Evidently, therefore, in the furnace, fineness, impurity and low-temperature fusibility will exist together and, hence, in sections of the fire, low-temperature fusion will result. Conceivably, a fairly high-fusion-ash coal as it exists in the bed might not be variable enough in the analysis of the ashes of its several parts to clinker badly, yet after segregation in mining, dumping, transportation, stocking and feeding to the furnace, it might develop these mass differences of analyses, so that large clinkers would form.

This study has a distinctly practical quality. It might be suggested, however, that if the separation were first by petrographic breaking followed by crushing, specific-gravity separation and size separation, the results might be even more striking than when the preliminary petrographic breaking to separate vitrain, clarain, durain, fusain and rocks of various kinds is omitted. However, such a more complicated determination might have less value than Dr. Gould's simple one. On the other hand, it might be urged that mining, dumping, transportation, stocking and feeding break the coal more in accord with the petrographic classes of the several coal components than does the more indiscriminate breaking in a crusher.

J. E. Tobey,† Cincinnati, Ohio.—The use of the ash-softening temperature, or the middle point, has served and is serving a valuable purpose in the selection of coal. We have long felt that sooner or later more consideration must be given the initial deformation point. We have seen many instances in the case of underfeed stokers where certain coals have had a long and satisfactory performance record wherein clinker trouble suddenly develops. In these cases it has been found that the ash-softening temperature and the fluid temperature have remained stable, but the initial deformation point has dropped a matter of 200°. In some of these cases it has been determined that foreign matter such as dirt or soil has been introduced into the coal, as occurs when storage yards are being cleaned.

It is assumed in these instances that sticky ash causes a growing or knitting effect on the fuel-bed materials, which in turn causes a blanketing of the tuyeres. This aggravating condition results in hot spots in the fuel bed, causing the formation of large clinkers, which disturb the fuel bed and result in the loss of stoker iron. In other words, with the ash-softening temperature normal, but with the initial deformation point depressed, fuel-bed disturbances do occur.

T. F. Downing, Jr.,‡ Philadelphia, Pa.—The authors have contributed a new direction toward the solving of a most difficult problem. However, many conditions must be considered before practical application of the method can be made. Different parts of the coal bed in a single mine face may differ materially in ash-fusion

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temperatures and widely in spread between initial and liquid fusions. The fact that some selection in buying takes advantage of such conditions would seem to indicate that some purchasers are going along the same lines as indicated in this paper.

As an illustration, consider some areas of central Pennsylvania, where beds carry two distinct benches. The top bench may be erratic in fusion limits but as much as 200° above those of the bottom bench. Usually these variations are reflected in difference of coal structure. The softening temperature of the total seam in a given mine may be 2700°F., that of 2-in. nut slack 2650° and that of ¼-in. slack 2400°. The crushing of the nut slack to slack does not always change the fusion materially. It is possible that had the authors screened their 2-in. nut slack and run separate tests on the natural 1-in. plus and the ¼-in. minus material their results would have shown a much narrower range of free fusible material in a given size and a wide spread as between sizes.

Another complication is the kind and amount of extraneous matter, bone, slate, fire clay, sandrock, or pyrite, with the coal. Unfortunately, one or more of these is usually present in coal shipments. Some of them are fusible and some refractory.

L. C. McCabe,* Urbana, Ill.—Significant differences in ash-softening temperatures are evident from investigation of banded ingredients of Illinois coals. Enough data are available to indicate that the vitrain and clarain ash are commonly low fusing and that fusain and durain ash are relatively high fusing (Table 11).

Table 11.—*Ash-softening Temperatures of Coal Components from Illinois Mines*

Component	Mine A	Mine B	Mine C
Vitrain	1810	1906	2023
		1918	1927
		2084	2260
Clarain	2130	2617	2073
			2206
Fusain	2270	2638	2565
			2732
			2349
Durain	2540		2732

While vitrain and clarain ash do not show marked differences in softening temperature by the standard method, the clinkers formed in an underfeed stoker during the combustion of these two types of coal are quite different. Fig. 1 shows pieces of the clinker rings of clarain and vitrain ash (mine A). The vitrain-ash clinker is dense and vitreous. The clinker from clarain combustion is porous; the granular particles reflect the size of the coal fired.

The grindability of the different types of coal from mine A are given in Table 12.

Durain has been found in relatively small amounts in No. 6 coal in southern Illinois, but wherever encountered it is difficult to grind. The sample on which grindability is reported contained 4 per cent ash, so the ash content does not appear to be a factor in the toughness of this type of coal.

* Associate Geologist, Coal Division, Illinois State Geological Survey.

TABLE 12.—*Grindability*
U. S. BUREAU OF MINES BALL-MILL METHOD

Component	Revolutions	Component	Revolutions
Vitrain	1084	Durain	3182
Clarain	1262	Fusain	413

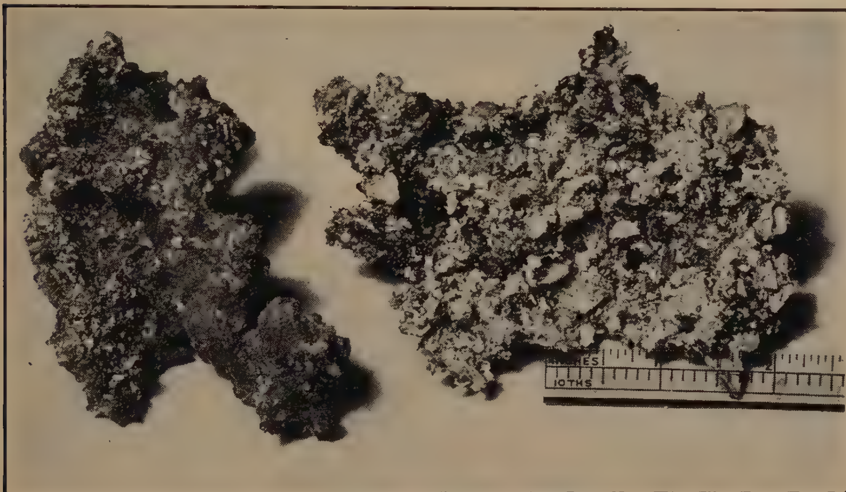

FIG. 1.—CLINKERS FROM STOKER.
Vitrain left, clarain right.

Friability follows the same pattern in the banded ingredients as grindability. Therefore, fusain is concentrated below 100 mesh, vitrain usually has its greatest concentration between ⅜ in. and 100 mesh, and clarain and durain, if present, are most abundant in the larger sizes.

These data bear out Gould and Brunjes in that intimate mixtures of coal ash are the exception during combustion. In coals of mixed type differences in friability, grindability and specific gravity complicate the ash-fusion problem.

Use of Pulverized Coal as Fuel for Open-hearth Furnaces Melting Steel for Castings

By Joseph P. Kittredge*

(Columbus Meeting, October 1939)

At the time this matter first came up in 1912, the National Malleable and Steel Castings Co. had seven basic-bottom open-hearth furnaces in its plant at Sharon, Pa., using fuel oil, then costing about 2½¢ per gallon. The fuel was satisfactory but its cost fluctuated widely, therefore the use of pulverized coal, which had been successful in the cement industry and in some metallurgical furnaces, was suggested.

After a thorough investigation, and an actual trial on one of the open-hearth furnaces, to be certain that the siliceous ash in the coal would not seriously affect the basic slags, the management of this company approved its adoption and a complete installation was started. The conclusion had been reached that the success of the operation depended on pulverizing the coal to greater fineness than had thus far been accomplished, and all equipment was chosen with this in mind. A coal drier was deemed essential, and one of ample capacity and efficiency was selected, which has done all that was expected of it, working without failure and with very small maintenance cost, and is still in operation.

Pulverizers

The company demanded a pulverization of at least 90 per cent through a 240-mesh screen, with consistent results, and a machine was chosen that was grinding other materials than coal to the fineness desired. The control of fineness was obtained through a vacuum separator, and was governed by the size of the separator and the number and construction of the valves necessary to expand and direct the air current. These valves can be set to obtain any degree of fineness desired, and the result is consistent.

Two machines were installed, each capable of producing 2½ tons per hour, 95 per cent through 240 mesh. These machines have done everything expected of them, have run continuously, and are still in operation.

An examination of some of our pulverized coal by elutriation and microscopic measurement disclosed that a 1-lb. lump of bituminous coal,

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* National Malleable and Steel Castings Co., Sharon Pa.

about 3 in. in diameter, having a total surface of about one-quarter of a square foot, came out with the surface of the particles exposing 3745 square feet.

Bins and Burner

Great care was taken in the design and construction of the bins to prevent the coal from hanging. The bins were made absolutely smooth inside, all rivets flush, and the downtake pipe was made 9 in. in diameter at the bin and 11 in. in diameter at the lower end above the cutoff gate.

The coal is fed to the burner by a double-pitch screw conveyor, which is driven by a variable-speed motor, giving the operator very flexible control of his fuel supply. A burner was designed (Fig. 1) in

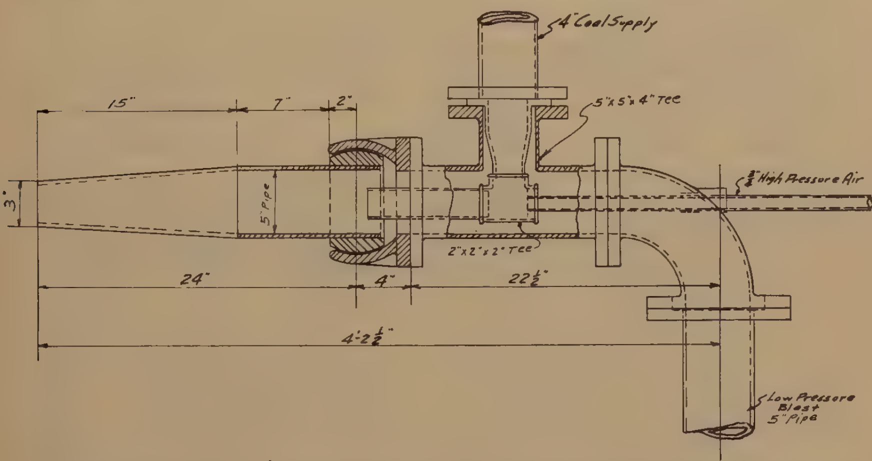

Fig. 1.—Burner for pulverized coal in open-hearth furnaces, National Malleable and Steel Castings Company.

which the coal drops by gravity from the screw into a 2 by 2 by 2-in. tee, from which a 2-in. pipe leads into the nozzle. The other leg of the tee is bushed to a ⅜-in. opening to admit 80 lb. of compressed air, which does the atomizing. Around the tee connection in the 5-in. casing is a 16-oz. blast coming from a constant-pressure, varying-volume, centrifugal blower. This line has a gate, which can be regulated by the melter. The nozzle has a universal joint, which permits the melter to direct his flame. The coal starts to burn a few inches from the nozzle, and has the appearance of a gas flame. The flame in the furnace should be short, not extending beyond the middle to two-thirds the length of the bath.

Slag and Ash

There is more slag produced in a furnace fired by pulverized coal than with oil fuel. It is primarily silicate of iron hence a removable slag

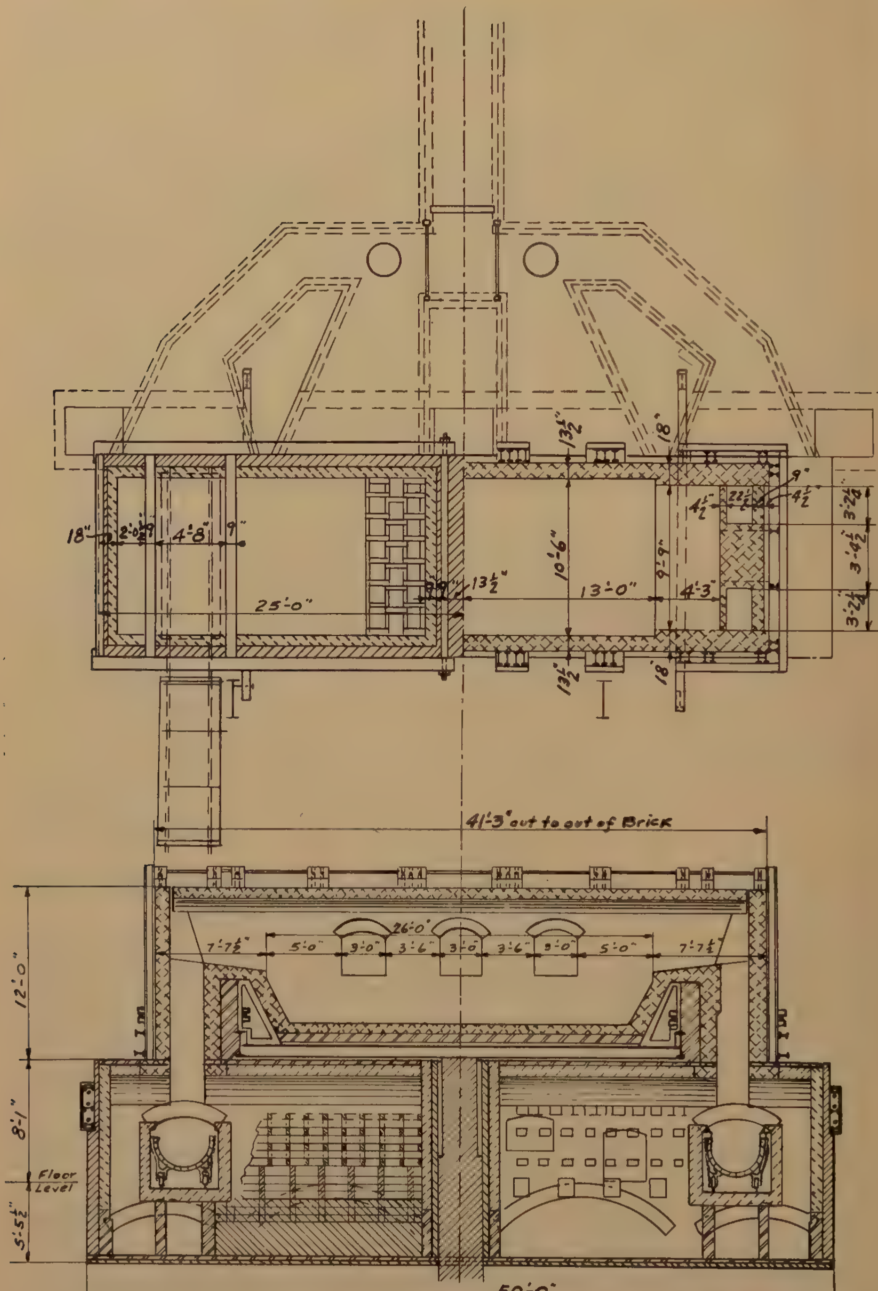

FIG. 2.—PLAN AND ELEVATION OF OPEN-HEARTH FURNACE, SHOWING REMOVABLE SLAG POCKET AND ARRANGEMENT OF CHECKERS.

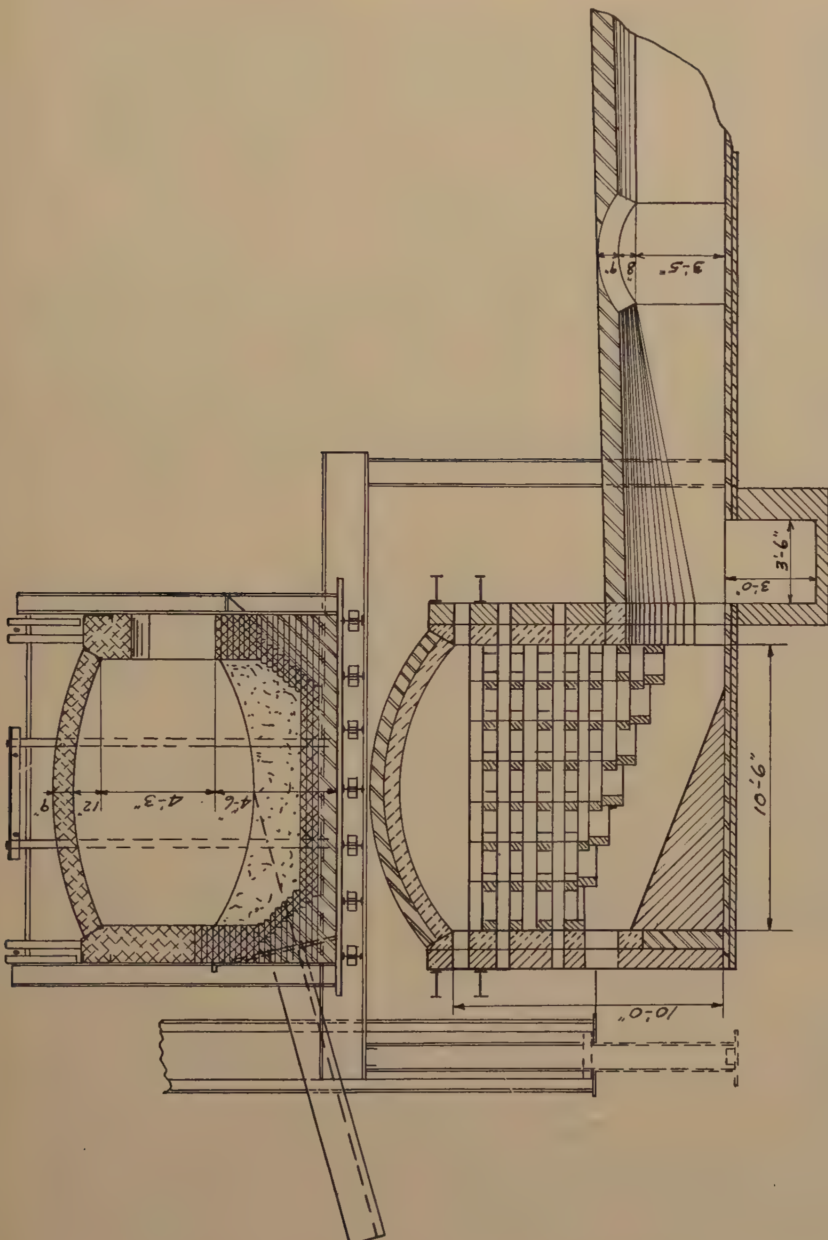

FIG. 3.—CROSS SECTION OF OPEN-HEARTH FURNACE SHOWING ARRANGEMENT OF CHECKERS, BOTTOM SLOPE AND PIT TO CATCH HEAVY DUST.

pocket is essential (Fig. 2). The crude but simple arrangement can be removed while the furnace is in operation, usually about once or twice a week, and only 15 min. is required to change one of the pockets. In further explanation of the removable slag pocket, there is a layer of damp sand packed in the bottom and sides of the pocket, to facilitate the removal of the slag without harming the brick lining. The ends of the pocket, of course, are bricked up when the pocket is in place. A slag pocket holds from 5 to 7 tons of slag.

In the designing of the checker layout (Fig. 3) of the furnace, it was planned to have all passages direct and as large as possible. The bottom of the checker chamber was sloped toward the exit flue, and openings were provided in the chamber walls so that the checkers could be examined daily and if any ash dust started to accumulate it was blown out with a high-pressure air pipe. It has been our experience that if this is done the checkers will outlast the life of the upper structure, which averages 200 heats.

With oil fuel 9-in. checker brick was used, exposing 9800 sq. ft. of radiating surface per chamber, but for pulverized coal 13½ by 2½ by 6-in. tile was installed. The openings going directly down were 8 by 11½ in., and the radiating surface only 5680 sq. ft. per chamber. This is only 58 per cent of that used with oil fuel, but the heats were made as rapidly as with oil fuel, if not more rapidly. A waste-heat boiler picked up the heat loss, making the total saving greater. We also eliminated the usual butterfly valve for reversing the furnace and substituted cast-iron dampers, as offering a more direct flow of the effluent gases carrying the ash. Later checker brick of special shape, 18 by 2½ by 16 in., were adopted. These have their upper edges chamfered to shed dust, and are set with a full bearing of the brick that interlocks, making a substantial construction. The openings are 12 by 12 inches.

Safety

Pulverized coal ground to the fineness cited above flows like water. All openings and joints in the pulverizer house should be tight, and bins should be very smooth inside and designed so that the coal will feed without hanging. While there has never been an explosion in the 26 years of experience by this company, there have been several inflammations caused by the coal hanging in the bins and feeding out below. When the coal dropped it pushed a large amount of air mixed with coal dust out of the safety vent above the burner, which hung like a black cloud under the roof of the building and on several occasions took fire, burning like a slow-running wave and setting fire to the window casings.

When the No. 1 plant was shut down, two of the bins over the open hearths were inadvertently left full of pulverized coal, which remained

there until the furnaces were torn down about 10 years later. There was no evidence of spontaneous combustion, nor had there been any appreciable deterioration of the coal; it was removed and used in the system of No. 2 plant.

Coal Used

Pennsylvania gas coal, used at first, analyzed as an average: volatile and moisture, 34.87 per cent; fixed carbon, 58.25; ash, 4.5 to 7.9; sulphur, 0.90 to 1.05. Later some coal from the Elkhorn district, Kentucky, was brought in, which averaged: volatile, 38.74 per cent; fixed carbon, 58.31; ash, 2.85 to 4; sulphur, 0.57 to 0.75.

With the increase in freight rates, Pennsylvania coal became the most economical fuel, and is now being used exclusively. Through better preparation and washing, coal varying from 5 to 6 per cent ash and about 0.80 sulphur is obtained.

Costs

Some time ago an extensive analysis was worked up by the company's accountants, giving a comparative analysis of costs with pulverized coal fuel in the open-hearth furnaces versus pulverized coal atomized with

Cost of Pulverized Coal Delivered to Open-hearth Furnaces, Based on Four Years Operation

Coal-conveying Department	Cost per Ton of Coal	Coal-pulverizing Department	Cost per Ton of Coal
Labor	$0.05	Labor operating coal-pulverizing apparatus	$0.47
Supply and tools	0.03	Coal for drier furnace	0.02
Repairs	0.07	Supply, tools and implements	0.03
Power	0.03	Repairs to permanent equipment	0.31
Depreciation	0.04	Power	0.67
		Depreciation	0.31
	0.22		1.81
		Cost of pulverizing and delivering coal to furnaces	$2.03

	Pulverized Coal	Oil Fuel
Net metallic charge, net tons	33,156.83	33,156.83
Heats run in this calculation	1026	1026
Fuel consumed per net ton of metallic charge	474 lb. coal	51.5 gal. oil

Cost of Pulverized Coal Delivered to Open-hearth Furnaces, Based on Four Years Operation.—(Continued)

	Actual Cost per Ton Net Metallic Charge	Estimated Cost per Ton Net Metallic Charge
Labor melting	$0.77	$0.77
Skull	0.06	0.06
Loading stock	0.35	0.35
Cleaning checkers	0.05	0.05
General	0.28	0.28
	$1.51	$1.51
Ladle fuel oil	$0.07	$0.07
Fluorspar	0.05	0.05
Limestone	0.30	0.30
Other department supply	0.01	0.01
	$0.43	$0.43
Open-hearth furnace repairs:		
Labor repairing furnaces	$0.50	$0.39
Brick	0.64	0.62
Chrome ore	0.10	0.08
Magnesite	0.28	0.15
All other furnace-bottom refractories	0.32	0.22
All other furnace repair materials	0.11	0.06
	$1.95	$1.52
Total open-hearth melting dept. less melting fuel	$4.45	$4.02

	Pulverized Coal	50-50 Basis Pulverized Coal and Natural Gas	Oil Fuel
Net metallic charge, net tons	33,156.83	33,156.83	33,156.83
Fuel consumed per net ton of metallic charge	474 lb.	2750 cu. ft. @ 33¢ per M cu. ft., 196 lb. coal	51.5 gal.
Fuel price delivered	$4.32[a]	$3.09	$0.0357[b]
Total open-hearth melting dept. less fuel and cost of preparing and delivering	$4.45	$4.23	$4.02
Cost of pulverizing	0.48	0.23	
Compressed air	0.07		0.07
Cost of fuel: coal	1.03	0.30	
Cost of fuel: gas		0.91	
Cost of fuel: oil			1.84
Cost of fuel delivered to the open hearths	$1.58	$1.44	$2.04
Total open-hearth melting dept. cost including fuel and delivery	$6.02	$5.67	$6.06

[a] Present coal cost delivered is $3.74.

[b] Present oil cost delivered is $0.03625.

high-pressure natural gas versus fuel oil, and extracts from this analysis are given below as a relative comparison today.

Production

The production of open-hearth steel for modern requirements is most exacting. Porosity in castings will not be tolerated by the trade; hence the steel must be killed, but overkilled steel is just as objectionable through failure of physical characteristics to meet the specifications. A growing amount of alloy steel is required. All of the open-hearth steel produced by the Sharon plant since 1913 has been melted with pulverized coal; 800,000 tons of steel have been melted in the No. 2 plant, with approximately 190,000 tons of pulverized coal and although the figures for the No. 1 plant are not available, a safe estimate for that plant would be 1,000,000 tons melted in all and, with annealing furnaces included, 300,000 to 400,000 tons of pulverized coal burned.

Conclusions

The most essential single feature is the proper preparation of the coal, which should not be less than 90 per cent through a 240-mesh screen, and preferably should be 95 per cent or higher through 240 mesh. The pulverizing plant should be prepared to produce such results consistently.

Melters and first helpers should be carefully trained to properly use this fuel. They will fear the intensely short, hot flame and wish to soften it to save the roof, as they reason. It is evident that the less coal is burned, the less ash there is to combat.

Removable slag pockets are essential.

A regular inspection to keep checkers and flues clean will prevent the accumulation of a drift of ash, which will stop the draft and cause the ash to accumulate very rapidly.

Over 20 years ago there was much interest and activity in the use of pulverized coal in open-hearth furnaces; much money was spent and many installations were put in, but these were all abandoned. Possibly manufacturers of equipment were responsible for the failure to develop this fuel successfully, by assuming that their systems, which had been successful in the cement and various metallurgical operations such as annealing and iron-melting furnaces, could be applied as readily to an open hearth.

With proper engineering study to perfect the equipment, and adapt open-hearth furnace design to the use of pulverized coal as fuel, I believe that economies far exceeding what we have obtained are possible, with marked economy to the steel manufacturer, a stimulation to the coal producers, and a great advantage to the railroads, all of which are conditions now sorely needed in this country.

DISCUSSION

(J. E. Tobey presiding)

W. Mittendorf,* Cincinnati, Ohio.—It would be interesting to know whether the coal consumption per ton of steel melted was less when using the low-ash high-heat southeastern Kentucky coals in comparison with the Pennsylvania coals tested. Also, whether there was any gas analysis made to check the carbon dioxide content against the air as measured from the blower.

R. F. Stilwell,† Columbus, Ohio.—What are the chief operating problems encountered in burning pulverized coal in open-hearth furnaces for melting steel castings as contrasted to the problems in burning pulverized coal in malleable melting furnaces where it has been the usual fuel for many years?

R. B. Textor,‡ Cleveland, Ohio.—In the use of pulverized coal as fuel for open-hearth furnaces, what is the relative effect of sulphur on the bath as compared with that in the use of gas or oil as fuel?

R. M. Hardgrove,§ New York, N. Y.—We agree that high fineness is essential in application of this nature and we have confirmed this fact in the application of pulverized coal to small furnaces, which would not have been possible if the coal had not been pulverized to 93 to 95 per cent through 200 mesh.

Direct firing has been widely adopted in the firing of cement kilns in recent years with better uniformity in combustion as well as simplified equipment and with eastern coals having low moisture, we believe it is applicable to open-hearth furnaces and that future developments will be along these lines. The use of pulverized coal in open-hearth furnaces offers a large field for use of coal and the pulverizing equipment is now developed to meet these conditions.

C. F. Herington,|| Pittsburgh, Pa.—Although pulverized coal was applied to open-hearth furnaces before 1914, it was not until the war period that any appreciable number of installations were made. The unusual war conditions appeared to call for a more efficient use of coal and it also was desired to use the available producer gas, natural gas, and oil for other purposes when necessary. The questions of design and proportion with respect to furnaces, checker chambers, ports, flues, waste-heat boilers and other details were incident to the problem of applying pulverized coal to open-hearth furnaces, and often were difficult to solve under the pressure of the production demands of the period.

However, a considerable number of steel plants—recognizing the opportunity for fuel economy involved—found it desirable to make installations of pulverized coal for melting even though skeptical of the probability that the incidental problems could be solved under the existing conditions. As far as the speaker knows, none of these installations are now in operation. In fact, it is believed that the installation described by Mr. Kittredge is the only existing example of what might be accomplished in melting steel with pulverized coal.

The principal reasons for the abandonment of the use of pulverized coal at the other melting shops mentioned were: (1) reduced costs of all fuels following the war period; (2) physical difficulties and lack of exact knowledge of furnace and regenerator

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design to accommodate pulverized coal; (3) lack of exact knowledge of the design of pulverized-coal feeders and burners; (4) lack of efficient pulverizing equipment that would reduce coals to the fineness necessary for rapid complete combustion; (5) metallurgical considerations—sulphur, ash, etc.

Even though the steel-melting shops abandoned pulverized coal in the early nineteen twenties, other industries took hold of the problem. Pulverized coal enjoys wide use in copper smelting, steel heating, malleable-iron melting and other operations. In fact, since 1922 approximately 90 per cent of the malleable-iron melting capacity in air furnaces has gone over to pulverized coal. And, of course, the use of this fuel in boilers is widespread.

As might be anticipated, the continued interest in pulverized coal, and the willingness of these industries to accommodate their furnaces to its use, to gain its advantages, permitted the pulverizer and equipment manufacturers to improve their designs and constructions to their present advanced state.

Today we are not confronted with the feeder, burner, and pulverizer difficulties that hampered the pioneers of 1918. Feeders and burners have been simplified and made foolproof, and efficient pulverizers are available that will reduce coals—with reasonable power and maintenance costs—until 85 per cent or more will pass through a 200-mesh screen.

Mr. Kittredge's paper is timely and will be found most interesting to steel melters. Steel plants are being modernized all along the line and it seems reasonable to believe that many melting shops, particularly in the smaller plants, might be well advised to consider the possibilities of pulverized coal under present conditions. Coals of less than one per cent sulphur are available and often freight costs that in other circumstances might be considered excessive can be absorbed in this operation.

J. P. Kittredge (author's reply).—Answering Mr. Stilwell's question: All open-hearth furnaces are reverberatory, using checker chambers. It is important to generate all the heat possible while the flame is passing over the bath, and not continue the burning in the checker chambers; therefore, the finer the coal, the shorter the flame, with less coal used and less ash to combat. While I also think the same principle would aid malleable melting, its need is not as imperative, because it is not necessary to combat ash in checkers, so possibly the higher cost of pulverization would not be justified.

I do not have available supporting data to answer Mr. Mittendorf's questions. My recollection, however, is that somewhat less Kentucky coal was used per ton of steel melted, but only in proportion to the lower ash content of the Kentucky coal. The lower ash and sulphur content of this coal, and its uniformity, determined its use. CO_2 readings were taken, although the data are not now available, but as the effluent gases were used through a waste-heat boiler, this point was not important.

Regarding Mr. Textor's query: Our experience showed that there was no more deleterious effect from sulphur in the use of pulverized coal than with gas or oil; in fact, we think that there was even a little less, if anything.

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(NOTE: In this index the names of authors of papers and discussions and of men referred to are printed in SMALL CAPITALS, and the titles of papers in *italics*.)

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